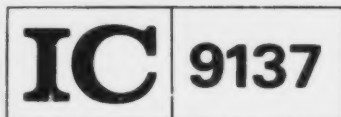


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Bureau of Mines Information Circular/1987

Eastern Coal Mine Geomechanics

Proceedings: Bureau of Mines Technology Transfer
Seminar, Pittsburgh, PA, November 19, 1986

Compiled by Staff, Bureau of Mines

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UNITED STATES DEPARTMENT OF THE INTERIOR

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UNITED STATES DEPARTMENT OF THE INTERIOR

Donald Paul Hodel, Secretary

BUREAU OF MINES

Robert C. Horton, Director

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PREFACE

The papers contained in this Information Circular address the results of several research studies conducted by the Bureau of Mines in an effort to improve mine roof support systems and mine design. Through eastern coal mine geomechanics research, new technology is being developed that will enhance mine productivity, improve resource recovery, and increase mine workers' health and safety.

The seven papers were presented at a technology transfer seminar on eastern coal mine geomechanics in November 1986. Technology transfer seminars are used by the Bureau of Mines to keep the minerals industry apprised of new technology and developments resulting from its research endeavors. Further information about current research efforts in other areas of concern may be obtained by writing the Bureau of Mines, Branch of Technology Transfer, 2401 E St., NW., Washington, DC 20241.

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UNIT OF MEASURE ABBREVIATIONS USED IN THESE PAPERS

deg	degree	lb	pound
ft	foot	lb/cu ft	pound per cubic foot
ft ²	square foot	m	meter
ft/d	foot per day	min	minute
ft/ft	foot per foot	mm	millimeter
gal	gallon	us/ft	microsecond per foot
gal/min	gallon per minute	pct	percent
gal/(min·acre)	gallon per minute per acre	psi	pound per square inch
gal/yr	gallon per year	psi/ft	pound per square inch per foot
gal/(acre·yr)	gallon per acre per year	psig	pound per square inch, gauge
h	hour	st	short ton
in	inch	yr	year
in/yr	inch per year		

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EASTERN COAL MINE GEOMECHANICS

**Proceedings: Bureau of Mines Technology Transfer
Seminar, Pittsburgh, PA, November 19, 1986**

Compiled by Staff, Bureau of Mines

ABSTRACT

The Bureau of Mines is establishing design criteria for effective roof support systems and is developing technology to improve mine planning and design. Geologic studies, surface and underground mine monitoring, and laboratory evaluations were conducted. The seven papers in this proceedings Information Circular present results of several investigations. Addressed are problems existing in both longwall and multiple-seam mines. Discussions include the effect of dewatering at a longwall mine site in the Pittsburgh coal seam and the effect of chain pillars on subsidence. Other results are measurement and characterization of longwall rock mass movement and an integrated design for stability in multiple-seam mining.

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INTRODUCTION

The Bureau of Mines conducted a research study of coal mine geomechanics in the Eastern United States. The goal is to improve mine planning and design by establishing better roof and ground support systems. Longwall mine roof failures often occur because the support system provides inadequate ground control at the face. Better design, selection, and operation of such support systems may help decrease the capital risk involved in equipping a longwall face. Proper supports will also enhance productivity, maximize coal recovery, and minimize health and safety hazards.

Problems in controlling ground are also prevalent in multiple-seam mining, which is practiced almost exclusively in the East. Past disregard for the sequence of mining multiple seams has led to major problems. Entry and pillar instability are caused by subsidence, pressure concentrations, and stress zones. If mining proceeds without concern for adjacent coal seams, our coal resources will become depleted at a much faster rate. New technology will provide high recovery of multiple-seam coal resources in an

economic and safe manner. To help alleviate some of these problems, the Bureau conducted a series of studies. The objective was to evaluate not only the effects of the mining method, but also how geology relates to support requirements.

A rock mechanics investigation monitored deformation of near-seam strata above a longwall panel in the Pittsburgh Coalbed. The study provided data concerning the caving mechanism associated with longwall extractions. Analysis of the data provided a better understanding of the interaction of strata behavior with longwall face supports. Another Bureau research study analyzed subsidence over mining operations. The purpose was to provide mine operators with a means of predicting movement of the surface ground and its effect on ground water. This research led to a computer model capable of predicting subsidence over longwall panels in the Northern Appalachian coal region.

The report presents further details about the results of these research efforts and of other studies.

STATE-OF-THE ART TESTING AND ANALYSIS OF MINE ROOF SUPPORT SYSTEMS

By Thomas M. Barczak¹

ABSTRACT

The Bureau of Mines is conducting research to evaluate the interaction of mine roof supports with the surrounding strata to provide for more effective design and utilization of these structures. Scientific testing of a system requires consistent definition of pertinent parameters and concepts. Various interpretations of support capacity, as they exist in the mining community, are briefly discussed in this paper, as are the deficiencies in historical uniaxial testing and analysis of mine roof support systems. Critical load analyses show that structural failure of the support can

occur at loads well below those of rated support capacity. The significance of support stiffness and post yield characteristics demonstrates the need to evaluate supports and the strata as a system rather than independently. Efforts to develop supports as load-sensing devices and strata activity monitors are described. The benefits and limitations of finite element modeling as an alternative to physical testing are discussed. A description of the Bureau's Mine Roof Simulator Facility and an overview of future research efforts are also provided.

INTRODUCTION

The development of more effective roof support systems and strata control strategies requires an accurate understanding of the interaction of roof support systems with the geological environment in which they are employed. Improvements in roof support design and strata control strategies can be realized only if the response of the support system and the conditions under which it is loaded are understood. To achieve this understanding, the Bureau of Mines has constructed facilities to evaluate full-scale roof support systems under controlled loading

conditions. Modern roof support systems, such as longwall shields, represent sophisticated structures, which require more indepth analysis than the simple prop supports of the past. State-of-the-art testing and analyses of mine roof support systems at the Bureau of Mines include laboratory, field, and theoretical studies using the most sophisticated equipment and analysis techniques available. This paper describes the Bureau's mine roof support facilities and research.

SUPPORT CAPACITY AND WHAT IT MEANS

One of the most fundamental measures of support performance is its load-bearing capability, which in the most elementary analysis is often ambiguously referred to as the capacity of the support. Several measures have been used to describe support capacity, and it is necessary to understand the differences and deviations of these measures in order to properly

interpret the load-bearing capability of the support.

One method of describing capacity for active supports, such as longwall shields, is the yield load of the hydraulic leg cylinders. In the case of shield supports, the effectiveness of the legs to support a roof load is diminished by inclination of the leg cylinder from the normal to the plane of the canopy; only the vertical component of the leg force should be used to determine the capacity

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of the support. Capacity in this context is more precisely defined as roof-to-floor or vertical support capacity, being premised by two inherent assumptions: (1) The support is generally assumed to be uniaxially loaded with no face-to-waste (horizontal) or parallel-to-the-face (lateral) loading, and (2) vertical load application is assumed to occur at the location of the leg reaction. Kinetically, there are components other than the leg cylinders that contribute to support capacity in structures such as longwall supports, and the actual capacity of the support should be determined from the total contribution of all components. Load applications at locations other than that of the leg reaction reduce the capacity of the support by introducing moments. The presence of off-axis (horizontal or lateral) loading can either increase or decrease vertical support capacity, depending upon the geometry and kinematics of the support structure.

Another method used to describe support capacity is termed "yield load density," which is defined as the yield load of the support per unit area of supported roof. This measure should be considered only as an average of load density reacted by the support and not in the context of a uniform pressure distribution of applied roof loading. It is computed simply by dividing the yield capacity by the area of supported roof. Since the yield capacity is determined from a concentrated load (or the equivalent stress

distribution) acting at the resultant support reaction, assuming a uniform load distribution acting on the canopy surface of magnitude equal to the yield load density will produce a support reaction in excess of the yield capacity. This means that support requirements should not be predicted simply from the weight of rock masses, assuming uniform load distribution on the support elements or load concentrations consistent with the center of gravity of the rock mass. The loading mechanism and interaction of the supports with the strata environment are discussed in a later section.

Finally, it must be recognized that supports are rarely loaded uniaxially. While the primary function of the support is to control roof-to-floor strata convergence, supports such as longwall shields are designed to control face-to-waste strata activity as well. Face-to-waste capacity of shield-type supports is generally rated at 30 pct of the rated vertical capacity, derived from an assumed 0.30 coefficient of the friction between steel and coal measure strata. The 30 pct of vertical capacity rating simply assumes that horizontal loads cannot be generated in excess of 30 pct of vertical loading since strata would then slip on the canopy interface. Actual capacity of the support to resist horizontal loading may be substantially higher. Unlike vertical loading, which is relieved by hydraulic yielding, horizontal load capacity is determined by the structural strength of support components.

CRITICAL LOAD ANALYSES

As indicated in the previous section, support capacity is generally determined from load concentration at the canopy resultant reaction. Support capacity will be reached prematurely by application of loads of substantially smaller magnitude at locations other than the resultant support reaction. For example, while a 500-st support could sustain a 500-st load concentrated at the leg (resultant) reaction, a load of less than 175 st applied at the tip of the canopy would

cause a typical two-legged shield support to yield.

Hydraulic yield capacity is not the only measure of a support's load-bearing capability. Loading conditions exist that could structurally damage the support with loads of magnitude well below the rated yield capacity. Critical load analyses are used to identify load conditions that cause maximum stress concentrations in the support structure. Figure 1 illustrates symmetrical load

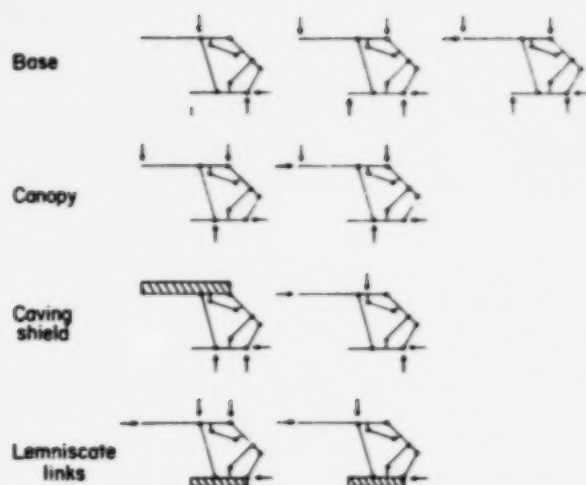


FIGURE 1.—Critical load test configurations.

conditions that would cause maximum stresses in each major component of a generic, lemniscate shield. Several partial-contact conditions are illustrated in the figure by arrows, which represent load applications or restraint of the structure. Full-contact conditions are

illustrated by a rectangle on the support member. Table 1 provides a description of the loading mechanism cases as well as a physical interpretation of the mine environment that could cause these load conditions.

The results of these critical load tests for one particular shield, representative of two-legged shields in general, are described in the following paragraphs.

CANOPY

The canopy is the least stiff of all shield components and is therefore subject to critical bending stresses at relatively small loads. Critical contact configurations suggest shield capacities of approximately one-half of rated hydraulic yield capacity. In other words, loads of less than half of the rated hydraulic yield capacity will cause stress concentrations approaching the yield strength of the steel under the contact conditions identified in table 1.

TABLE 1. - Critical load tests

Shield component	Loading mechanism	Physical interpretation
Base.....	Create maximum bending stress in base by supporting base as a simple beam with canopy inducing concentrated force at center span of base.	Base supported on toe and rear with leg force causing bending of base structure.
Canopy.....	Same mechanism as for base with bending created in canopy.	Same physical description as for base with canopy simply supported at tip and hinge.
Caving shield.....	Simulate horizontal displacement of canopy relative to base by creating horizontal couple as shown in figure 1.	Canopy and base promoting horizontal reactions due to vertical shield convergence.
Component link.....	Same strategy as with caving shield. Maximum loading occurs when canopy rotation is prevented, causing shield reactions at hinge pin and in caving shield.	Do.
Tension link.....	Same strategy as with caving shield. Maximum tension link loading occurs when links are in horizontal configuration (low shield heights).	Do.

BASE

The base is the stiffest of all shield components, and no contact configurations were found to induce critical stresses for this particular shield.

CAVING SHIELD

Roof-to-floor convergence of the shield, which is thought to be the predominant loading mechanism in an underground environment, produces relatively little loading in the caving shield. However, horizontal displacement of the canopy coupled with a horizontal restraint of the base to cause relative motion between the canopy and base creates stresses approaching yield strength of the caving shield material with vertical loads below the hydraulic yield capacity of the support. The effect of gob loading directly on the caving

shield was not investigated in this study.

LEMNISCATE LINKS

No contact condition was found to induce more than 6,000 psi of stress, which is less than 20 pct of the material yield strength. Apparently much of the energy is absorbed by the leg cylinders and in caving shield bending, with little energy being transferred to the lemniscate links. The primary function of the links under these test conditions is simply as a guidance system to control canopy canopy motion.

It should be noted that these test results are for one shield only. Other shields may behave differently depending on their structural characteristics, but the critical load conditions should be applicable to most shields of this basic configuration.

INTERACTION OF THE SUPPORT WITH THE STRATA

In the evaluation of roof support systems it is important to understand that the roof support elements and the surrounding strata act as a system, responding to changes in the physical mine environment due to extraction of coal and associated redistribution of stresses. The strength (capacity) of a roof support element alone is not a meaningful measure of support performance. The loading of a roof support element can be significantly dependent upon the stiffness of the support structure, as a stiffer structure will receive a higher load from converging mine strata than will a softer structure. This concept is illustrated in figure 2, where two hypothetical structures of different stiffness, K_1 and K_2 , with $K_1 < K_2$, are displaced an equal amount. As shown, the resultant load on the stiffer structure would be 1.5 times that on the softer structure under the same load condition.

The significance of support stiffness for behavior of the support is most critical in passive roof support systems, such as posts and cribbing. Recently, concrete and steel-fiber-reinforced (SFR) concrete have been used as substitute

materials for wood in crib applications. The higher compressive strength of concrete compared to wood (about eight times) results in a much higher yield strength (capacity) for cribs constructed of concrete. However, as seen from the load-displacement characteristic of wood and concrete illustrated in figure 3, concrete produces a much stiffer crib structure than does wood. These results indicate that a solid SFR concrete crib

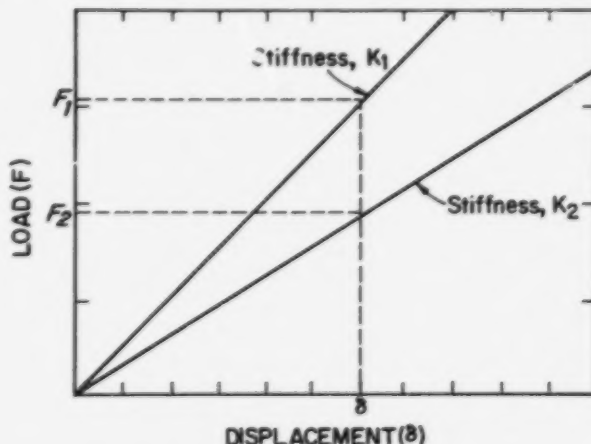


FIGURE 2.—Significance of support stiffness.

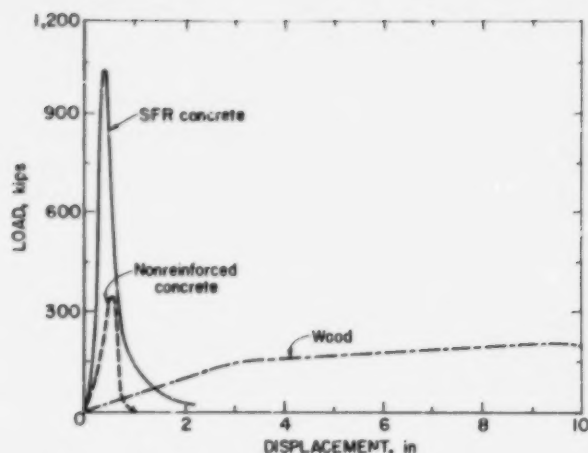


FIGURE 3.—Crib loading characteristics.

would experience a load 18 times that of a wood crib subjected to the same displacement prior to reaching yield strength of the crib. The displacement at which the yield strength of the support is reached is also a critical measure of support performance. The stiff concrete supports, despite having a much greater yield strength, reach yield load at much smaller displacements than cribs constructed of wood. Tests indicate concrete cribs fail at less than 0.5 in of displacement, whereas wood can displace nearly 3 in before reaching yield load and over 10 in before failing. Therefore, if the convergence of the roof is irresistible (displacement loading), the high yield strength of concrete cribs

will be of no advantage if the roof convergence exceeds 0.5 in.

Equally important to the loading characteristics of passive roof support systems is postyield behavior. As seen in figure 3, the stiff concrete supports lose load-carrying capability almost immediately after reaching yield load, whereas wood continues to provide support resistance for several inches of displacement. In terms of energy absorption, the wood crib can absorb more energy than a concrete crib owing to its ability to deform and maintain resistance to load. The idealized roof support should be initially stiff to be rapidly load bearing, should have yield strength sufficient to support the deadweight response of the immediate fractured strata, and should have sufficient postyield characteristics to be compatible with the displacement behavior of the overburden.

The stiffness of an active roof support, such as a longwall shield, is also critical to the interaction of the support with the strata. A stiff structure that generates high reactive forces can cause unnecessary fracturing of incompetent roof or floor strata or cause further instability of unstable strata. While the hydraulic support is self-relieving, the yield load of modern powered supports ranges from 250 to 800 st; this is sufficient to cause pressure distributions on the canopy and base that could damage incompetent roof or floor strata.

SUPPORTS AS MONITORS

Roof support elements, both passive and active structures, can be used as roof load monitors by proper instrumentation of the support structures. Past efforts to use roof supports as load-sensing devices have generally been limited to one-dimensional analysis by summation of measured forces to obtain a vertical (roof-to-floor) reaction. As previously indicated, supports are usually not loaded uniaxially. Modern longwall supports, such as the shield, are designed to resist both roof-to-floor (vertical) and face-to-waste (horizontal) loading,

requiring two-dimensional analyses to determine vertical and horizontal load reactions. The Bureau has demonstrated that vertical and horizontal support resistance of a longwall shield can be reasonably determined from static rigid body analyses by measurement of leg, canopy capsule, and compression lemniscate link forces. The two-dimensional model assumes no gob loading, nor does it recognize any out-of-plane loading or the effect of moment loading due to imbalances in the leg and lemniscate link forces. From controlled tests in the Bureau's

Mine Roof Simulator, vertical shield loading could be predicted to within 3 to 5 pct and horizontal loading to within 25 pct.²

In theory, the limitations of the two-dimensional model can be overcome by three-dimensional modeling of the support structure to account for out-of-plane forces and moments. Unfortunately, the advancement of a three-dimensional model is difficult since the accumulation of unknowns far outpaces the available force and moment equilibrium equations, making the system statistically indeterminate. While elimination of unknowns with reasonable engineering judgments as to non-participating forces produces a solution, the shield support is primarily designed to resist loading in two dimensions (roof-to-floor and face-to-waste), making the two-dimensional model adequate for most analyses.

It is also recognized from tests with shield supports that vertical convergence produces both a vertical and a horizontal load reaction; likewise, a horizontal displacement produces both a horizontal and a vertical support reaction. In other words, the shield support not only reacts axial loads in direct response to strata displacements (shield convergence), but also induces offaxis load reactions as a result of the mechanics of the shield structure. Only load reactions resulting from strata activity need to be resisted for successful application of the longwall method; support-induced loads are indications of inefficient designs if they are of no benefit to strata control. To put this in perspective, in situ tests indicate that over 75 pct of horizontal load experienced by shield supports is the result of setting the support against the roof. There is also some evidence that a significant portion of the remaining 25 pct

is due to vertical roof convergence and not face-to-waste strata activity.

To resolve the issue of the source of horizontal shield loading and to develop methods whereby the supports can truly be used as monitors of strata activity, the Bureau is investigating the utilization of a linear elastic model with two degrees of freedom. Mathematically, the model is expressed as follows:

$$F_v = K_1 v + K_2 h \quad (1)$$

$$F_h = K_3 v + K_4 h \quad (2)$$

where

F_v = vertical support resultant load reaction,

F_h = horizontal support resultant load reaction,

v = vertical shield displacement,

h = horizontal shield displacement,

and

K_1, K_2, K_3, K_4 = stiffness coefficients.

By controlled uniaxial displacement tests in the Bureau's Mine Roof Simulator, the stiffness coefficients are determined for a particular shield. Using numerical values for the stiffness parameters, the inverse of equations 1 and 2 can be used to determine shield displacements, which are indicative of mine roof convergence and strata activity, if resultant shield loading is accurately known. This also enables horizontal loading produced by face-to-waste strata activity to be distinguished from shield-induced horizontal loading due to vertical roof convergence.

MATHEMATICAL MODELING--AN ALTERNATIVE TO PHYSICAL TESTING

Mathematical models provide a simple, cost-effective method to evaluate load responses of structures without the need

for full-scale physical testing. While simple rigid body statics may be used to determine the elementary behavior of a

²Barczak, T. M., and R. C. Garson. Technique To Measure Resultant Loading on Shield Supports. Paper in Rock Mechanics

in Productivity and Protection (Proc. 25th Symp. on Rock Mech.). Soc. Min. Eng., 1984, pp. 667-679.

support in relation to an applied load, more sophisticated analyses are required to analyze the load-displacement relationship of a support structure. The linear elastic model presented in the previous section is a first step in evaluating roof supports as elastic bodies.

One of the most powerful techniques used to analyze structures is finite element modeling. According to its basic principle, a structure is idealized as a composition of a number of finite pieces rather than continuous elements. This concept enables the step-by-step buildup of the load-displacement relationship of a structure as a whole from those basic elements of which the structure is composed. The accuracy of the model is dependent upon the number of elements as the stresses are averaged over the area of the elements. Proper selection of elements and boundary conditions is required because different types of elements behave differently. Knowing the geometry and elastic properties of the elements allows computation of structural deformations for each of the finite elements from which areas of critical stress concentrations can be identified.

The Bureau has used finite element analyses in the following ways in mine roof support research:

Identification of load cases for testing. - The critical load tests presented in the section on critical load analyses were determined from a simple two-dimensional finite element model.

Location of strain gauges for physical testing. - It is generally desirable to locate strain-measuring instrumentation at areas of critical stress concentrations.

Identification of critical stress concentrations. - Three-dimensional models have been used to identify critical stress concentrations in major support components.

Evaluation of canopy pressure distribution from strain contours on the canopy structure. - Measurement of canopy pressure distribution by analysis of underside strain contours proved difficult

owing to multiple contact conditions that produce similar strain profiles and to difficulty in modeling partial contact conditions.

Finite element modeling of supports has proved more challenging than originally anticipated. The general behavior of the structure in terms of load distribution can be determined from finite element modeling, but an accurate picture of load transfer in the shield structure is sometimes difficult to obtain. Mathematical modeling requires an accurate definition of the problem, including both the properties of the structure and the load and boundary conditions. Discrepancies in finite element model predictions are often caused by improper modeling of load conditions. This is particularly difficult in modeling support behavior because of the multitude of contact configurations that exist with rock strata interfaces. Longwall supports are basically crude pin-jointed structures, and modeling pin frictions in joints can be difficult. Factors such as friction, ability to accurately model internal plating, and the effect of stress concentrations at discontinuities can have a significant impact on results.

The Bureau's experience with finite element modeling of longwall shields is that simple beam models constructed to fit a known shield response from physical testing are more successful and much cheaper than complex models, which may geometrically look more like the support structure, but are very difficult to construct to properly simulate the structural fabrication of the shield.

A further implication of the Bureau's research in finite element modeling of roof supports is the discovery that complicated shield component constructions are sensitive to some unique load applications. While current supports are sufficiently oversized that these unique load conditions do not represent problems, efforts to improve designs from a stress optimization viewpoint will require more refined load analyses than the relatively simple criteria presently used in support design.

BUREAU OF MINES FACILITIES

The heart of the Bureau's roof support test facilities is the Mine Roof Simulator (MRS) illustrated in figure 4. The MRS is unique in the world in its abilities to apply both a vertical and a horizontal load simultaneously.

Both the vertical and horizontal axis can be independently programmed to operate in either force or displacement control. This capability permits tests such as true friction-free controlled loading of shields, which cannot be accomplished in uniaxial test machines since the shield reacts a horizontal load to vertical roof convergence. Friction-free tests of this nature can be accomplished in the MRS by allowing the platen to float in the horizontal axis by commanding a zero horizontal load condition. Likewise, the MRS can apply controlled horizontal loading to a shield support, whereas uniaxial test machines can apply only vertical loading with no control over horizontal load reactions or capability to provide a specified horizontal load to the structure.

The machine incorporates 20-ft-square platens with a 16-ft vertical opening, enabling full-scale testing of longwall support structures. Capacity of the simulator is 1,500 st of vertical force and 800 st of horizontal force and controlled displacement ranges of 24 in vertically and 16 in horizontally. Load and displacement control is provided in four ranges operating under a 12-bit analog-to-digital closed-loop control network, providing a load control capability of better than 0.1 kip (100 lb) and displacement control capability of better than 0.001 kip in the smallest load-displacement range.

Machine control and data acquisition are achieved by a DEC 11/34 computer. Eighty-eight channels of test article

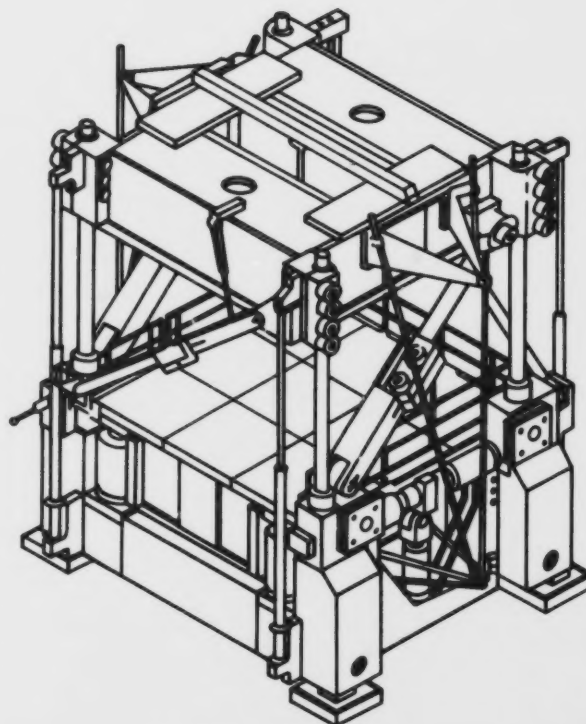


FIGURE 4.—Mine roof simulator.

transducer conditioning are provided. Data acquisition is interfaced with the control network so that machine behavior can be controlled by response of the test article instrumentation. For example, tests can be terminated or held when strain values reach a designated level in specified areas of the support structure. High-speed data acquisition is available with a separate DEC 11/23 computer at a rate of 300 samples per second. An X-Y-Y recorder provides real-time plotting of three data channels, and all data are stored on computer disks for subsequent processing and analysis.

FUTURE DEVELOPMENTS AND CONCLUSIONS

The Bureau of Mines plans continued research on mine roof support systems in an effort to develop more cost effective and safer systems for the mining industry.

Several aspects of ground control, support design, and strata interaction are not yet well understood. The Bureau hopes to advance the state-of-the-art of

ground control and support testing with the following goals:

1. Determine the effectiveness and limitations of finite element analysis in support design and behavior. Recommend guidelines for advanced analysis of mine roof support systems using finite element analysis.

2. Continue efforts to develop techniques to effectively use supports as strata monitors to develop a more basic understanding of strata activity so that support requirements can be more closely engineered to strata behavior.

3. Continue to explore and propose concepts of support behavior, such as the significance of support stiffness in load behavior, which may not be apparent to mine operators, support manufacturers, or other researchers.

4. Continue to explore advanced technology applications to current and novel mine roof support concepts or support designs.

In conclusion, ground control is critical to successful mining operations. Nearly all of the financial risk associated with a mining venture can be attributed to ground control. Current mining operations require capital-intensive systems costing several million dollars for each installation, and roof support fatalities continue to be the number one killer. Testing and analysis of roof support systems is necessary to provide better utilization of support systems and the development of improved designs for the future.

CHARACTERIZATION AND MEASUREMENT OF LONGWALL ROCK MASS MOVEMENT

By Jeffrey M. Listak,¹ John L. Hill III,¹ and Joseph C. Zelanko¹

ABSTRACT

The Bureau of Mines has conducted a rock mechanics study to monitor deformation of near-seam strata above a longwall panel in the Pittsburgh Coalbed. The primary goal of this research was to determine the height of caving immediately behind advancing longwall face supports. This study, although site specific, provides information on the caving mechanism associated with longwall extractions so that strata behavior and its interaction with longwall face supports can be better understood.

Two vertical boreholes, positioned 550 ft apart along the centerline of a longwall panel, were drilled from the surface to intercept the coalbed at a depth of approximately 650 ft. Various downhole

geotechnical instruments were used to monitor strata deformation. In addition, surface elevation surveys were conducted to differentiate between surface and subsurface activity.

This report discusses the caving characteristics of the strata as the longwall panel approached and passed beneath the boreholes. Physical property data are also presented to demonstrate the relationship between caving behavior and local geology. Data show that immediate caving of strata above the longwall face occurred at a height of less than 23.5 ft and that strata behavior above longwall extractions is highly dependent upon lithology, with major disturbances occurring at weak lithologic zones.

INTRODUCTION

For a safe and productive longwall operation, the optimization of both roof control and operational efficiency is essential. However, these two major contributors to longwall success are dependent upon the accurate prediction of roof support capacity requirements. Longwall failures of the past have been attributed to lack of understanding of roof and floor behavior and poor caving characteristics in the gob area.

Historically, roof caving in the gob has been evaluated by several methods. One method is a theoretical approach based on the cantilever beam theory of structural mechanics. This approach assumes that the stratified roof rock layers act as cantilever beams above the extracted coal. By knowing the material properties and the thickness of the layers, the extent of the interlayer loading

can be determined (1).² As the roof layers fail in stepwise fashion, caved material swells or bulks to fill the void and the caving height can be calculated.

Rock classification is another method used to characterize mine roof for caveability prediction. Much of this work has been done in the European coalfields by Pawlowicz, Kidybinski, and Kostyk of Poland and by Proyavkin and Davidyanc of the U.S.S.R. as cited by Kidybinski (2). In addition, Ghose (3) used geotechnical logs to classify mine roof for cave prediction in coal mines in India. Ghose also cited the notable classification systems proposed by Vasiliev et al. and Nenasheva et al. of the U.S.S.R. and the Roof Quality Index of Bilinski and Konopko of Poland. Bieniawski (4) developed an engineering classification of

¹Mining engineer, Pittsburgh Research Center, Bureau of Mines, Pittsburgh, PA.

²Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

rock masses called the Geomechanics Classification or rock mass rating (RMR). This classification system utilizes six parameters, which are measurable in the field or which can be obtained from borehole data, to determine the "stand-up time" for an unsupported roof span.

More recently, numerical modeling has been used to provide good estimates for cave prediction (e.g., the finite element method, distinct element method, and boundary element method).

Although these methods are of value since they offer guidance, precise identification of the caving behavior of the strata is still unresolved. Very few systematic studies that actually monitored the caving mechanism associated with longwall mining have been conducted. Field investigations (5-7) performed to directly measure bed separation over longwall extractions have had limited success. However, one such study conducted recently (5) used a borehole caliper tool to measure bed separation and had conclusive results that are very similar to the findings stated later in this report.

To develop a better understanding of longwall strata behavior and its interaction with longwall face supports, the Bureau of Mines installed various geotechnical instruments in two boreholes drilled over a longwall extraction located in southwestern Pennsylvania. This study is intended to lay the groundwork for additional research in order to develop a data base for strata behavior above longwall extractions. The information from these studies could effectively improve the method for selecting longwall roof support capacities.

Several analytical methods (8-10) have been developed to predict longwall strata behavior and associated face support loading. These methods for selecting the proper load capacity of longwall roof supports are based on the assumption that some finite volume of roof material, often assumed to be cubic or parallelepiped in geometry, is being held by the support. The boundaries of this volume of material are defined by the spacing of the supports (width), the supporting distance from face to gob, and the height and angle of caving. The height of caving is generally estimated as a factor times the height of extraction, the factor being determined by estimating the bulking factor of caved material. Formulae using assumed caving heights for predicting load density vary considerably. Wilson (8) assumes that caved rock occupies 1.5 times the volume of the same rock in situ and therefore maintains that the height of caving (the height of caving above the level of the roof) is twice the extraction height. Wade (9), however, assumes that caved material will occupy 1.25 times the volume of rock in situ, yielding a caving height equal to four times the extraction height.

To find the caving horizons above a longwall panel extraction, two instrumentation stations were installed in vertical boreholes located along the centerline of one longwall panel. Each station utilized instruments that measured both horizontal and vertical displacements as a function of longwall face advance. In addition, surface evaluation surveys were performed to differentiate between surface and subsurface displacements.

ACKNOWLEDGMENTS

The authors thank Rocrest, Inc.,³ Plattsburgh, NY, for cooperation in

designing, manufacturing, and installing the borehole instrumentation. Special thanks is extended to Girard Theroué and David Prentice of Rocrest, Inc., for their assistance at the field site.

³Reference to specific manufacturers does not imply endorsement by the Bureau of Mines.

DESCRIPTION OF TEST SITE

The longwall panel under investigation is located within the Appalachian Plateau Province of western Pennsylvania. Structural relief in the region does not exceed 350 ft, and dips are generally less than 4°. Mining takes place in the Pittsburgh Coalbed, which lies stratigraphically within the Pennsylvania age coal-bearing strata of the Monongahela Group (fig. 1). The longwall panel is mined to a height of 5.8 ft; overburden depths vary throughout the length of the panel. Figure 2 illustrates the lateral continuity of this interval over the panel under study. The panel (panel 3) dimensions are 630 ft wide and 5,570 ft long. This longwall panel utilizes a four-entry headgate and tailgate system with square pillars on 90-ft centers and entries 15 to 18 ft wide.

Roof support along the face was maintained by Dowty 460-st two-leg shield supports with setting pressures of 3,600 psi. The rate of advance for the longwall face was approximately 35 ft per three-production-shift day.

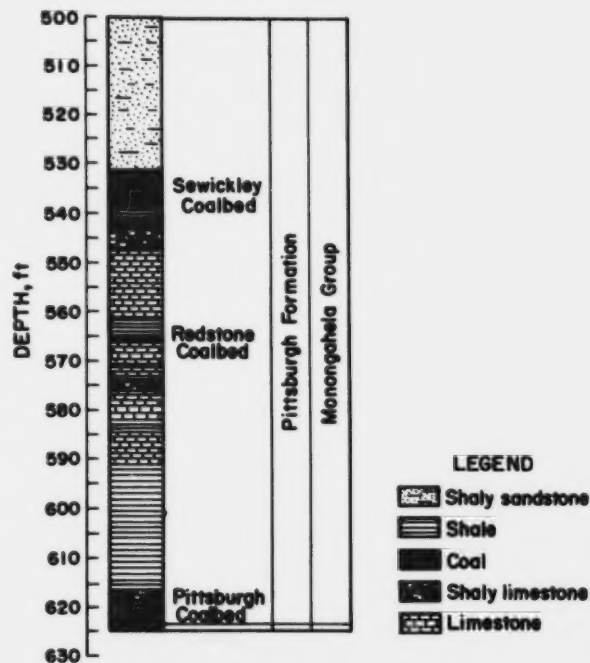


FIGURE 1.—Generalized stratigraphic column of the Monongahela Group.

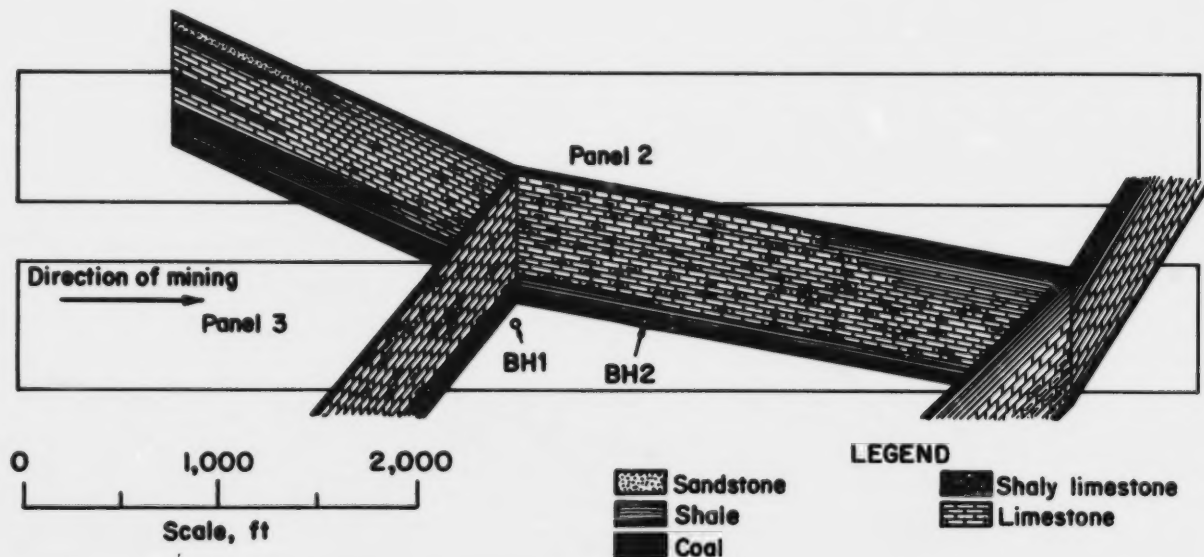


FIGURE 2.—Fence diagram of strata overlying the study panel.

SITE PREPARATION--BOREHOLE DRILLING

Factors considered during site selection included topography and terrain, surface rights (private land ownership), and environmental restrictions. From a technical standpoint, however, the most important consideration was to choose a test site that would yield representative results over the length of the panel. Therefore, in addition to the nontechnical considerations, borehole locations were chosen toward the center and along the length of the centerline of the panel to minimize the effects of the panel boundaries on the caving process. The two monitoring stations were located 2,600 and 3,150 ft from the start of the panel (fig. 3).

Two 6-in-diam boreholes were drilled through the coalbed. With the exception of a 15-ft standpipe at the surface, the holes were not cased. This allowed the borehole anchors to be set directly in distinct stratigraphic members. A combination of rotary and core drilling was used to drill the first borehole (BH1). One hundred feet of core was extracted from BH1 for descriptive geologic logging and laboratory testing. This interval included the entire interburden between the Sewickley and Pittsburgh Coalbeds. The surface at BH1 had a mean sea level elevation of 1,004.68 ft, and the borehole depth reached 630 ft. This final depth was approximately 3 ft below the base of the Pittsburgh Coalbed. Geophysical logging was performed on each hole to determine lithology and the location of water-bearing strata. These logs were also used to calculate a rock strength index.

The quality of the borehole is of utmost importance. It is imperative to have a straight hole that is free of obstructions so that problems during instrument installation can be kept to a

minimum. However, in this study, borehole conditions were not ideal. This prevented placement of borehole anchors at desired depths (5 ft above the coalbed), so caving of the immediate roof behind the longwall supports could not be detected. Figure 4 shows a borehole directional survey of BH1. Although the horizontal difference between the top and bottom of the borehole was only 3.5 ft, sharp deviations of the borehole, due to the spiraled drill path, prevented placement of a borehole anchor at the desired depth of 615 ft, which is 5 ft above the coalbed.

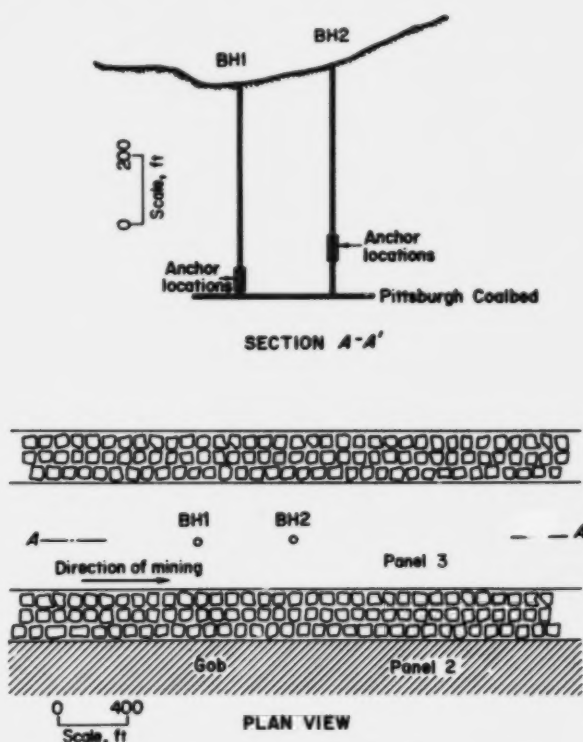


FIGURE 3.—Study panel and cross section of borehole anchor locations.

ROCK MASS CHARACTERIZATION

An assessment of geologic and mechanical rock properties of the entire unit of strata overlying the longwall panel at the mine site was conducted to relate these characteristics to caving height and lateral overburden movement. To accomplish this, geophysical logging was carried out over the entire length of each of the two boreholes.

Figure 5 is a summary of the characteristics obtained for the Pittsburgh and Sewickley interburden at the longwall site. The characteristics include geologic description, strength index from well logs (dynamic elastic modulus of deformation), uniaxial compressive strength, indirect tensile strength, and rock quality designation (RQD) (11). As the figure shows, the immediate roof rock (approximately 28 ft) above the Pittsburgh Coalbed is mainly composed of weak shales with a low RQD. Since the RQD values shown were calculated over each 10-ft core run, the three clayey shale units (indicated as "disintegration zones" in the figure) do not specifically stand out as weak zones. However, each of these units disintegrated upon removal from the core barrel. As will be discussed later, it is probable that caving height was coincident with one of these horizons.

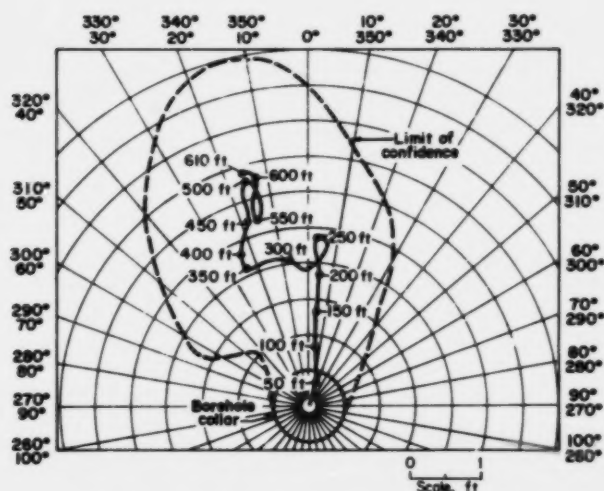


FIGURE 4.—Borehole directional survey of station 1.

American Society for Testing and Materials (ASTM) standards were used for strength evaluations of the NX core from BH1, including uniaxial compressive strength and Brazilian tensile strength (diametral compression) tests. All of the core used was ample in size to meet specimen standards, and a statistically significant number of specimens were tested for most of the rock types recovered. Mechanical property data could not be obtained for several areas in the borehole because either the core disintegrated during recovery or the lengths of recovered core were inadequate for the preparation of test specimens. These areas are indicated in figure 5.

After the hole was cored, the bottom 100 ft was reamed to 6 in for geophysical logging and subsequent installation of the borehole instruments. Unfortunately, by the time geophysical logging was conducted, the tools were unable to proceed beyond 16 ft above the coalbed.

The suite of geophysical tools used at the site included caliper, natural gamma, density, resistivity, spontaneous potential, temperature, fluid conductivity, and sonic logs. The sonic and density logs were used to calculate a strength index for the rock above the coal seam where the extensometer anchors were positioned.

The following equation was used to give a relative strength value for those sections of the overburden that could not be tested in the laboratory:

$$E_d = \frac{\rho b}{(\delta t)^2} \times 3.36 \times 10^9$$

where ρb = bulk density, g/cm³,

δt = interval transit time, μ s/ft,

and E_d = dynamic elastic modulus of deformation, psi.

This equation was developed by Schlumberger Well Services (12) and relates the sonic and density logs to the dynamic elastic modulus of deformation. The

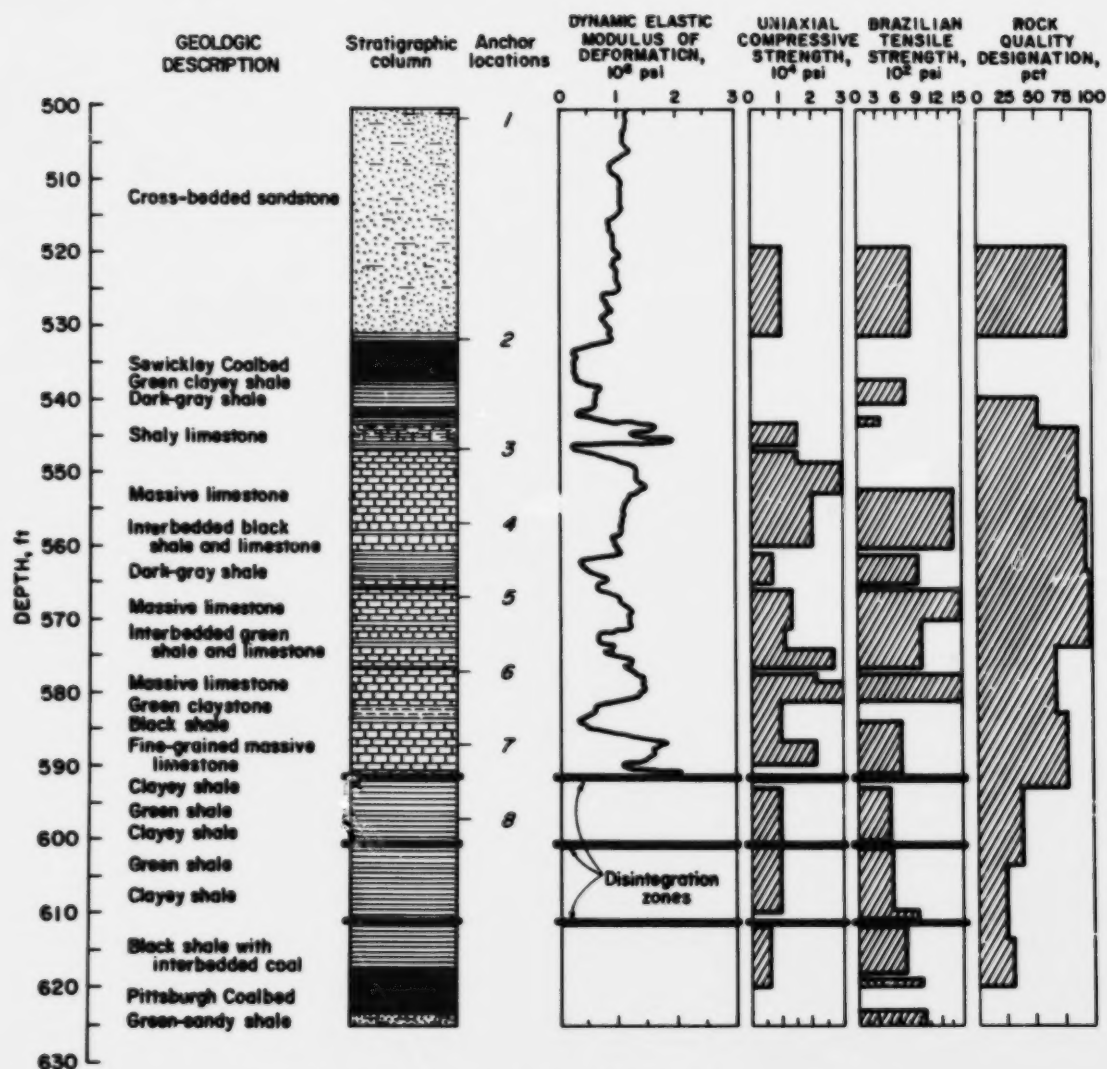


FIGURE 5.—Geologic and rock strength characterization of borehole 1.

value is not to be regarded as an absolute strength value but rather as an upper limit of the possible strength of the rock.

As compared with actual laboratory tests of specimens, the strength index showed a good correlation between low index values and corresponding low strength test values. For the higher strength test values, the strength index often indicated a relatively higher strength than did the strength value that resulted from laboratory tests. Bond (12) offers an explanation for this relationship:

... a competent appearing formation could be fractured enough to weaken the rock structurally but not enough to create an observed effect on the logs. On the other hand, formations appearing weak on the strength index curve could not be considered stronger than the calculated index. Therefore, the strength index should be considered an indication of the upper limit of bed competence.

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In addition to physical property testing of the immediate roof rock, in-mine geologic mapping was conducted in the gate roads of the monitored panel. All clastic dikes, slips, and roof falls were recorded as shown in figure 6. A high frequency of clastic dikes (often referred to as clay veins) is found throughout the study area. These dikes are characterized as normal fault-type fractures infilled with a clay matrix and inclusions of coal, sandstone, and shale. The dip of the normal fault-type fractures ranges from the vertical to 45° , with as much as 3.3 ft of vertical displacement along the fault plane. At this

site, the dikes had no obvious preferred orientation, although their relatively close spacing may have facilitated caving of the gob. Otherwise, observations showed that the dikes adversely affected ground control only in isolated areas.

Coal cleat measurements were also taken at the site. The mean orientations of the butt and face cleat were determined to be $N 25^\circ E$ and $N 65^\circ W$, respectively. The direction of mining was subparallel with the face cleat at $N 60^\circ W$. Although no joints were found within the roof rock, other studies have shown that coal cleat orientations often mirror the orientations of joints in overlying strata

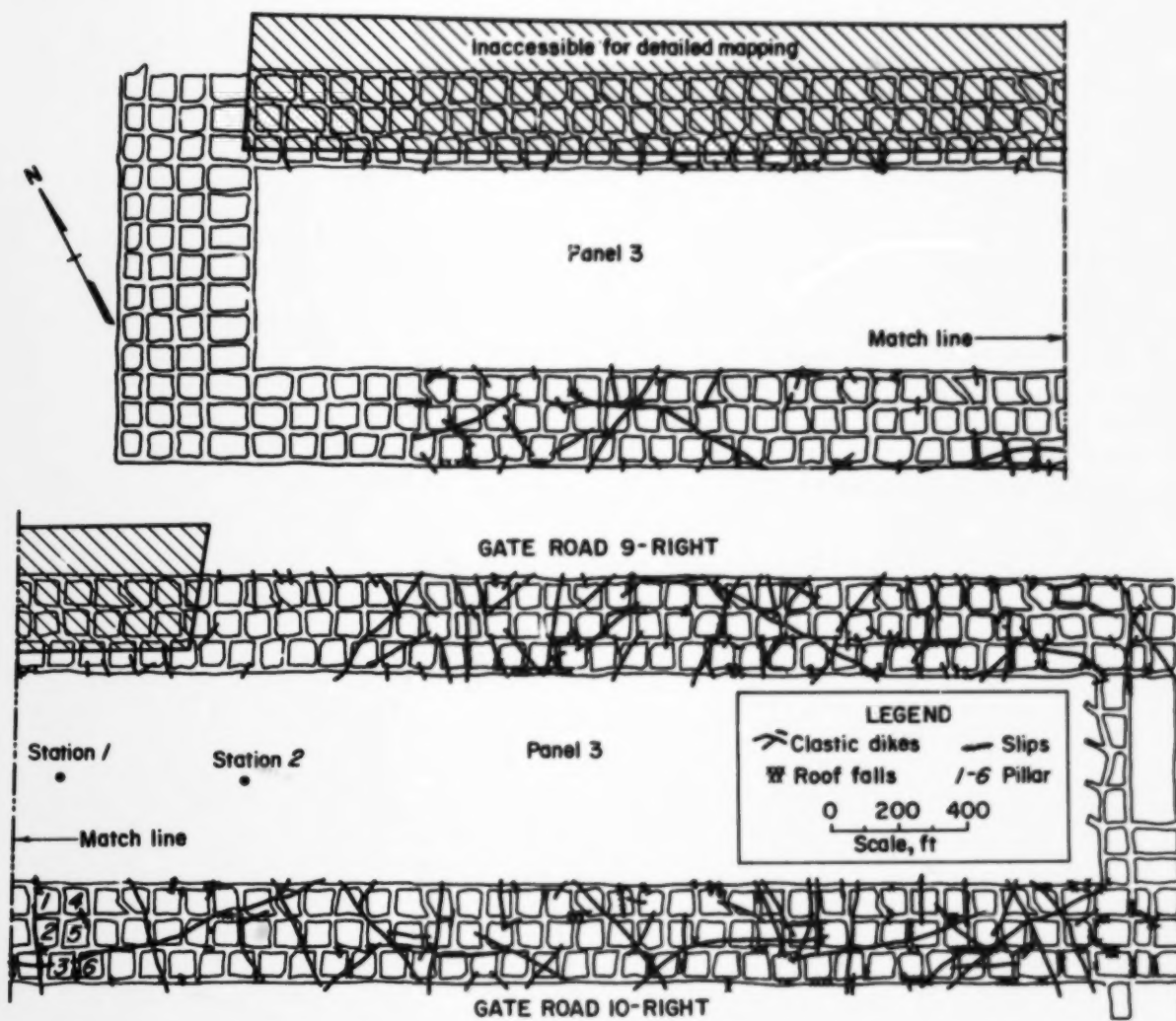


FIGURE 6.—Roof falls and geologic anomalies of gate road entries adjacent to study panel.

(11, 13-14). Thus, since the face line of the panel was subparallel with one of the major orientations of the coal cleat, it is possible that jointing in the overburden may have contributed to the caving characteristics of the gob.

Overall, the rock mass characterization supports the favorable longwall mining conditions that are evident at the mine.

INSTRUMENTATION

Each instrumentation site was comprised of a surface monitoring station and a subsurface instrument installation as shown in figure 7. A premining surface elevation survey was performed at each site to establish an elevation datum, and successive surveys took place at intervals during panel extraction. These surveys allowed differentiation between surface subsidence and subsurface strata activity. Each hole was instrumented with a multiple-point borehole extensometer and an inclinometer casing, which allowed measurement of vertical and horizontal displacements, respectively. Vertical measurements were made by direct readout of extensometer scales, continuous recording units connected to the extensometer system, and direct readout of a magnetic settlement probe system. Lateral displacements were calculated from inclinometer probe measurements.

MONITORING OF VERTICAL STRATA DEFORMATION

To measure vertical displacement of substrata, each borehole was equipped with a multiple-point borehole extensometer; this device detects vertical strata movement through the use of mechanical spring anchors. The anchors were positioned at specific intervals within the strata and connected to the surface by stainless steel wires. Depth intervals for anchor placement were selected following an analysis of the recovered core and the geophysical logs. Each anchor was positioned within a distinct stratigraphic member. Interfaces between stratigraphic units were avoided on the assumption that caving would be most likely to occur along these planes.

The immediate roof is strong enough to remain stable between the tip of the shield line and the face, and weak enough to allow immediate collapse directly behind the shield line. The weak, clayey zones of the immediate roof appear to allow for a consistent caving height, and, thus, consistent loading on the shields.

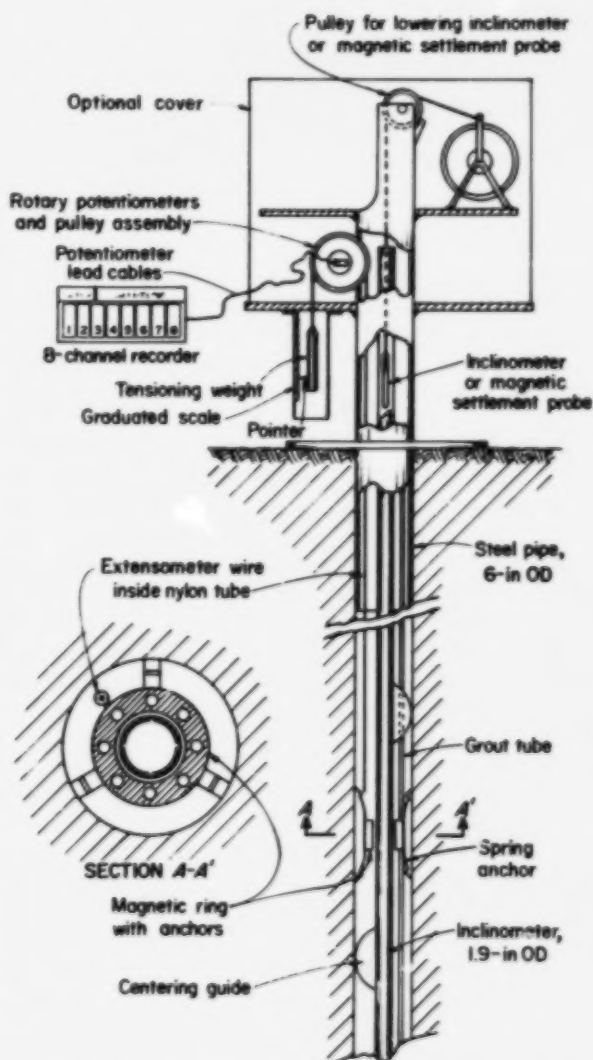


FIGURE 7.—Complete extensometer-inclinometer system. (Courtesy Roc-test, Inc., Plattsburgh, NY)

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Each of the two 6-in boreholes accommodated eight anchors, the maximum number that could be used in this borehole diameter. Eight sections of 1.9-in-OD polyvinyl chloride (PVC) inclinometer casing were prepared at the factory to accept the eight anchors. The remaining sections of inclinometer casing were standard 5-ft sections. The anchor springs were compressed and held closed during installation by nylon strings, which passed through the casing and were attached to opposite pairs of anchor springs (fig. 8). The anchors were positioned on the casing at the predetermined depths shown in figure 5. The 5-ft sections of casing were glued together and lowered down the hole. A grout tube was fastened to the first section of casing and lowered as the system was being assembled (fig. 9). Grouting of the borehole was necessary to seal any water-bearing zones, which could have caused water inflow into the mine when the borehole was undermined. The casing remained centered in the hole by means of two sets of PVC centering blades installed on the casing, 5 ft above and below each anchor

(fig. 10). A 1/16-in-diam stainless steel wire surrounded by 1/4-in-diam oil-filled nylon tubing was attached to each anchor. The tubing was necessary to allow free movement of the wire after the hole was grouted. A wire-tubing assembly was attached to each of the eight anchors and lowered with the anchor and casing assembly. The grout tubing and each of the wire-tubing assemblies were positioned on scaffolding and lowered into the borehole as the 5-ft sections of casing were added. When the entire assembly had been lowered into the borehole, the anchors were set in place by dropping a weighted knife down through the casing to cut the nylon strings.

A reference head was placed at the top of each borehole. The head consisted of a 6-in-OD steel pipe with a welded circular steel plate used to seat eight potentiometer-pulley assemblies (one for each anchor). Each anchor wire passed through the center of the instrument reference head, over its own pulley, and was fixed with a 50-lb tensioning weight. Graduated scales were fastened to the outside circumference of the head to allow direct



FIGURE 8.—Multiple-point borehole extensometer anchor positioned on inclinometer casing.

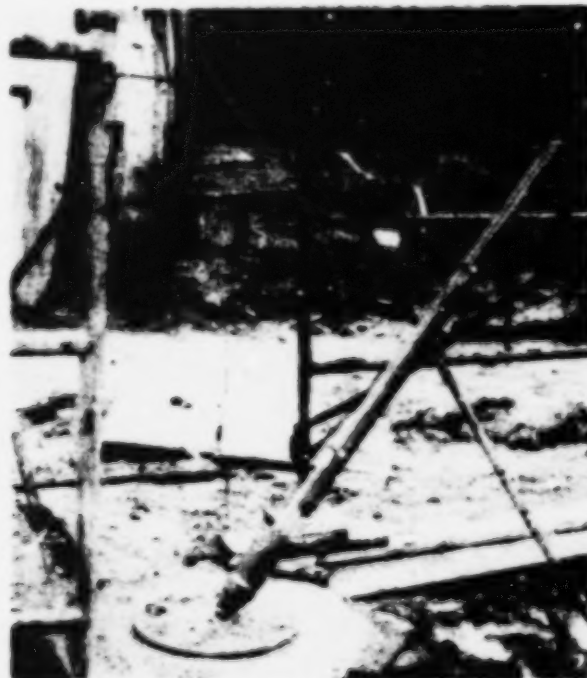


FIGURE 9.—Grout tube attached to lead section of inclinometer casing.



FIGURE 10.—Borehole anchor and PVC centering blades.

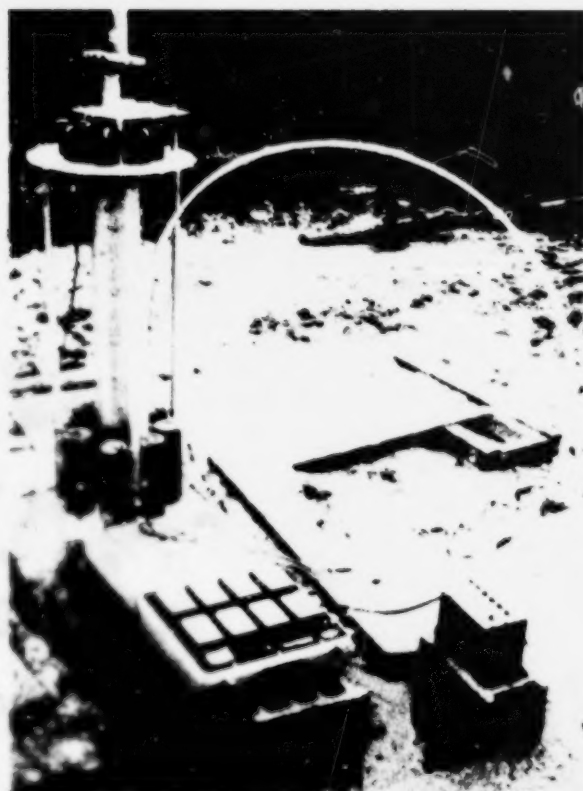


FIGURE 11.—Multiple-point borehole extensometer reference head.

readout of displacements. For remote readout, the potentiometer leads were soldered onto a terminal panel to which a continuous recorder was connected. The head also incorporated a large pulley for lowering the inclinometer¹⁸ and magnetic settlement probe (fig. 11).

The magnetic settlement probe, which works by magnetic inductance, was used to verify anchor locations in the strata (fig. 12). Magnetic rings were incorporated into each of the eight borehole anchors, creating a magnetic field inside the inclinometer casing. A reed switch probe was connected to a graduated cable mounted on a cable reel. An audible buzzer housed inside the cable reel was activated by entry of the probe into the localized magnetic field produced by the anchor. This device provided excellent results for verification of anchor locations. However, after mining progressed

beneath the borehole, high concentrations of methane began propagating up the borehole, preventing further use of the non-permissible magnetic probe and the continuous recording unit.

MONITORING OF HORIZONTAL STRATA DEFORMATION

An inclinometer probe was used to measure the progressive changes in the angle of inclination of the inclinometer casing. These measurements provided an evaluation of lateral movement as mining approached the station. The probe was supported laterally in the casing by guide wheels and suspended vertically by a cable connected to a reel and readout unit. The guide wheels traversed opposing longitudinal grooves spaced equally 90° around the inside circumference of the casing for directional control. Two

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servoaccelerometers, mounted with sensitive axes 90° apart, simultaneously monitored inclination both parallel and perpendicular to the direction of mining. Recording of data was accomplished by the use of a digital indicator equipped with a magnetic tape cassette recorder.

FIELD DATA ANALYSIS

Base reference data were established at each station approximately 500 ft in advance of mining, and readings were taken relative to the longwall face position. Surface elevation datums were also established for each borehole prior to mining. Instrument readings were taken weekly while the face was more than 200 ft from the stations, and daily when the face was within 200 ft of the instruments. Face advance was obtained from the mine daily, and surface elevation surveys were made

Although the inclinometer probe output is recorded in terms of the angle of inclination, lateral deflections can be calculated easily from these data. Figure 7 illustrates the complete instrument arrangement.

periodically after mining passed beneath the boreholes.

EXTENSOMETER DATA ANALYSIS

To establish the progress of caving, borehole anchors were positioned in the mine roof strata as shown in figure 5. The deepest anchor in BH1 was located 23.5 ft above the coal seam. A distance of 10 ft separated anchors 8 through 3, with anchors 2 and 1 separated by 15 and 30 ft, respectively.

It is important to note that the anchor displacements shown are with reference to the extensometer head located at the surface. The movements of the extensometer head were determined by surface elevation surveys. The displacements shown in figures 13 and 15 have not been corrected to include the measured movements of the surface.

Initial anchor movement was detected in all anchors when the face had approached to within 500 ft of station 1 (fig. 13). Since it cannot be associated with caving, this movement has been attributed to lateral displacements of strata and to a rising surface elevation in advance of the face (fig. 14). As the face drew near and passed beneath the station, anchor positions were recorded hourly based on the assumption that large movements would be seen immediately after the longwall supports passed beneath the borehole. However, anchor movement was not detected at that time. Figure 13 shows that significant movement (caving related) in anchors 2 to 8 began when the face was 35 ft past the station. Anchor 1 (the farthest from the extraction) began moving when the face was 75 ft past the station, at the same time that surface subsidence began (figs. 13-14). The abrupt failure of the immediate roof



FIGURE 12.—Magnetic settlement probe.

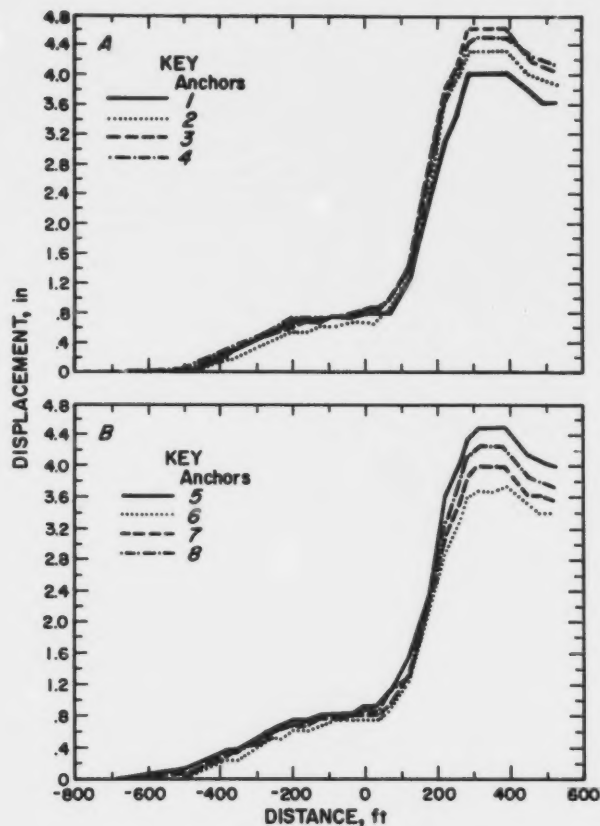


FIGURE 13.—Station 1 extensometer displacements.

associated with the advance of the long-wall supports was not detected by anchor 8, which was positioned 23.5 ft above the extraction. This indicates that 23.5 ft is above the upper limit of the first strata separation. Descriptive geologic logging of the immediate strata revealed three very weak bands (approximately 6 in thick) of soft clayey shale in the immediate 25 ft of roof. These weak zones occurred at heights of 8, 17, and 25 ft, and it is assumed that immediate caving behind the supports occurred up to one of these zones. Using these zones as possible caving horizons and relating each to the following equation yields three different bulking factors for the extraction height of 5.8 ft.

$$H = c + h$$

$$H = ck$$

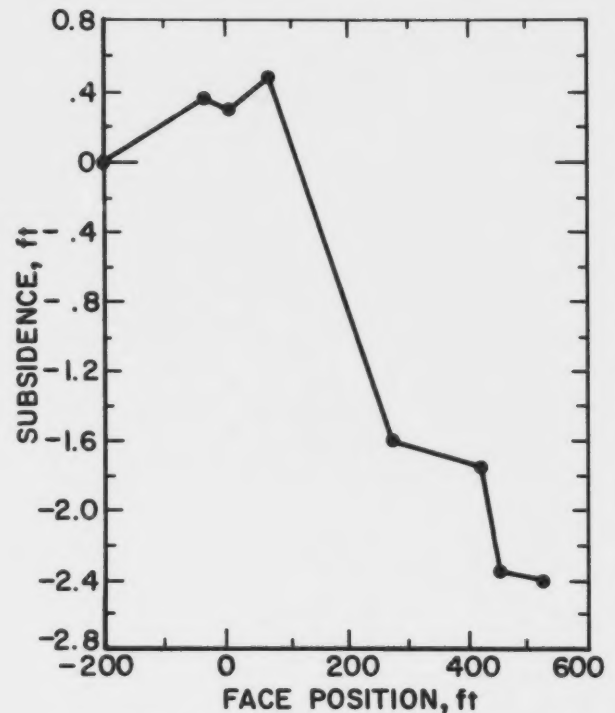


FIGURE 14.—Surface elevation survey data for station 1.

where h = extracted height, ft,

c = height of caved material from roof level of extracted height, ft,

H = distance from floor level to caving horizon, ft,

and k = bulking factor, unitless.

Substitution yields

$$k = \frac{h}{c} + 1.$$

The caving horizons of 8, 17, and 25 ft yield bulking factors of 1.72, 1.34, and 1.23, respectively. The fact that anchor 8 did not begin to detect small movements until the face had passed 35 ft beyond station 1 suggests that immediate caving behind the supports occurred at some height less than 23.5 ft. The bulking factor of 1.34 for the 17-ft horizon closely corresponds to the commonly used

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value of 1.33 for bulking of shale, which comprises 30 ft of the immediate roof. Therefore, caving is assumed to have occurred up to the 17-ft horizon. Surface subsidence and anchor movement occurred concurrently when the face was between 75 ft and 290 ft past the borehole (figs. 13-14). Maximum surface subsidence during this period was 2 ft, and maximum extensometer movement of anchor 8 was 4.25 in. The fact that both surface and small subsurface movements occurred at the same time and at a distance of 75 ft beyond station 1 suggests that the entire overburden member above the assumed caved height of 17 ft began to sag and compact the gob material.

Although the most subsidence occurred between 75 and 290 ft, surface movement did not cease until the face was 530 ft past the station. Anchor movement ceased at 290 ft. This difference is attributed to the closure of fractures in the strata above the anchors. Closure is seen as an apparent upward movement of anchors.

INCLINOMETER SURVEY DATA ANALYSIS

The inclinometer was used to detect lateral deflections in advance of the longwall face. The inclinometer probe measured the angle of tilt of the casing within the borehole in two directions, parallel and perpendicular to the centerline of the panel. The following analysis will refer to an A and B direction. The A direction is parallel to the direction of mining. Positive A deflections are movements toward the direction of mining, and negative deflections are in the opposite direction. The B direction refers to deflections perpendicular to the direction of mining. Positive deflections are movements toward the previously mined panel, and negative deflections show movement toward the adjacent unmined panel.

The inclinometer casing in borehole 1 was installed to a depth of 604 ft, which was 16 ft above the mined height of the Pittsburgh Coalbed. Initial readings were established 290 ft in advance of the approaching longwall face. Although the inclinometer casing was lowered to a depth of 604 ft, the initial readings

could only be taken to a depth of 489 ft. This blockage was attributed to the sharp deviation in direction of the borehole at a depth of 500 ft, as shown on the directional survey that was performed immediately after the hole was drilled (fig. 4).

Readings were taken daily as the face approached and passed beneath the monitoring station, but they had to be discontinued after the face had passed 5 ft beyond the borehole because high concentrations of methane were vented up the inclinometer casing. Inclinometer survey data show that the vertical strata movements discussed in the previous section were accompanied by lateral deflections of the strata. Figure 15 shows the variation of lateral deflection of borehole 1 relative to face advance. Figure 15A shows movement in the A direction (parallel to the direction of face advance), and figure 15B shows movement in the B direction (perpendicular to the direction of face advance). Figure 15 reveals that shear zones begin forming at various depths in the borehole 263 ft in advance of the face. Large deviations from the initial reading began to occur

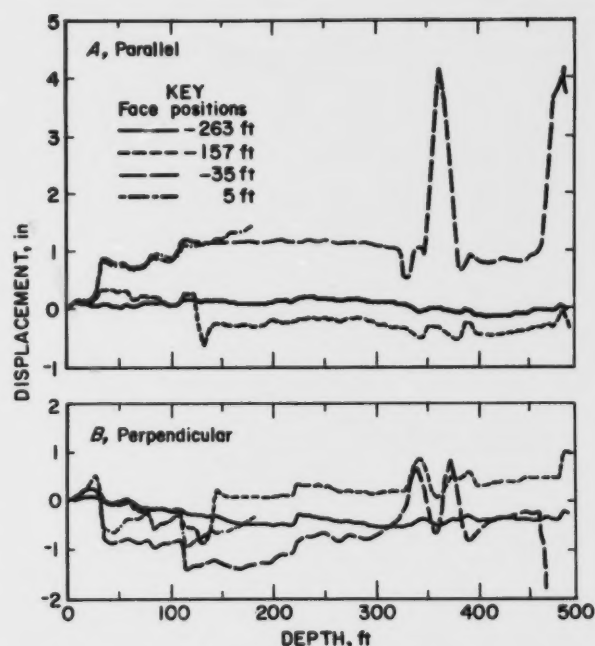


FIGURE 15.—Lateral displacements of borehole 1 parallel and perpendicular to the direction of face advance.

when the face was 157 ft from the borehole. In both the A and B directions, movement began to develop at a depth of 112 ft. Geophysical logs show a soft fire clay at this depth. Two other areas of activity at this face position occur at depths of 325 and 370 ft, where the strata again are composed of fire clay.

When the face advanced to within 35 ft of the borehole, greater movements were apparent. Three distinct areas are discernible. The aforementioned depths of 112, 325, and 370 ft show relatively large displacements (fig. 15). At 325 ft, a 15-ft member of fire clay exists.

A maximum deflection of 4.2 in can be seen at 360 and 480 ft. The shear zone begins to form at 460 ft in a 5-ft layer of carbonaceous shale that is situated between an upper member of sandstone and a lower member of limestone. When the face passed beneath the station and was 5 ft beyond the borehole, a shear zone developed at a depth of 179 ft, prohibiting the probe from progressing past this point. A final reading taken after face advance had progressed more than 195 ft past the station revealed that the highest progression of shear had extended to within 46 ft of the surface.

RESULTS

Significant anchor movement and surface subsidence was detected when the face had progressed 35 ft past station 1. Surface and subsurface movements continued to increase until the face had advanced to 290 ft past the station. At this point positive anchor deflection ceased. Surface subsidence continued slightly to a face position of 530 ft. Differential surface and subsurface displacement is attributed to the closing of fractures in the strata above the anchor positions. This appeared on the reference head as upward anchor movement.

Anchor 8 (the deepest) in station 1 did not detect an abrupt failure of the roof immediately after passage of the supports beneath the borehole but moved a total of 4.25 in as the face moved 310 ft past the station. This indicates that caving of the immediate roof occurred less than 23.5 ft above the top of the coalbed. Thus, the actual bulking factor must be greater than 1.25.

Three very weak bands of clayey shale are present between the top of the

coalbed and anchor 8. It is assumed that immediate caving occurred up to one of these weak horizons. Calculation of a bulking factor based on caving to the 17-ft horizon yields an estimated bulking factor of 1.34.

Inclinometer surveys show the formation of a number of shear zones throughout the length of the borehole. Shear zones were detected in station 1 263 ft in advance of the face. As the face drew nearer to the station, lateral displacements in the borehole became more apparent. A comparison of inclinometer survey data and geophysical logs revealed that shear zones are associated with weak strata horizons. These weak horizons occurred in fire clay material at depths of 46, 112, 325, and 370 ft. Another displacement of 480 ft occurred at a 5-ft layer of carbonaceous shale. As the face progressed beneath the borehole, a shear zone developed at a depth of 179 ft and prevented further inclinometer readings. The final shear zone detected was at a depth of 46 ft.

CONCLUSIONS

The primary goal of this investigation was to define the height of caving immediately behind the advancing longwall supports. Previous estimates of caving height, predict the height of caving to be two times (8) and four times (9) the extraction height. This investigation revealed that caving occurred less than

23.5 ft above the coalbed, most likely at a height coincident with one of three clayey shale zones. These zones are located 8, 17, and 25 ft above the coalbed, and calculation of bulking factors based on caving to each of these horizons yields values of 1.72, 1.34, and 1.23, respectively. The fact that 1.34 closely

corresponds to the commonly used bulking factor for shale (1.33) suggests that the caving horizon occurred at 17 ft, or three times the extraction height. Results of a field study by Matthews (5) revealed similar results.

Inclinometer data revealed another characteristic of longwall strata behavior. Comparison of inclinometer data with geophysical logs showed that the

major lateral deflections occurred in weak strata (i.e., fire clay and carbonaceous shale).

Based on these observations, the behavior of strata over longwall panels appears to be largely dependent upon lithology. Future studies should allow the caving behavior of various lithologies to be characterized.

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MULTIPLE-SEAM MINING PROBLEMS IN THE EASTERN UNITED STATES

By Gregory J. Chekan,¹ Rudy J. Matetic,¹ and James A. Galek²

ABSTRACT

The Bureau of Mines, in an effort to improve planning and development in coal mining, is currently investigating strata interactions associated with mining of multiple coalbeds. Strata interactions between adjacent coalbeds occur frequently in the Appalachian coalfields and can have unfavorable effects on both product cost and worker safety. Two common interactions that occur between adjacent coalbeds are subsidence and pillar load transfer. At two mine sites where such ground interactions were present, the Bureau conducted geologic studies and gathered various geotechnical information on pillar and entry stability using rock mechanics instrumentation. At the mine affected by pillar load transfer, the results of the study show: (1) overburden depth changed dramatically in the study area and reached a maximum at the study site, (2) innerburden thickness was less than a pillar width (40 to 45 ft),

(3) heaving was experienced in both mines where sandstone and/or shale floor units were observed, (4) overlays of the mine layout show pillars were not totally superpositioned, (5) a maximum of 5 in of roof-to-floor convergence was measured, and (6) monitoring of pillar pressures showed that only the pillar core was loading, an indication of a stiff pillar approaching failure.

At the mine affected by subsidence, measurements show that the undermining had little effect on upper mine pillar stability, but had a more severe effect on the development and maintenance of entries. Roof-to-floor measurements recorded over four times more convergence in entries developed over gob than in entries developed over support pillars in the lower mine. This has led to major roof falls in the active panels, resulting in production delays and supplemental roof support costs.

INTRODUCTION

Simultaneous mining of adjacent coalbeds or mining over or under a previously mined-out coalbed occurs frequently in the Appalachian Region of the Eastern United States. In West Virginia, Pennsylvania, and Ohio, it is estimated that 57 billion st of coal exists in a multiple-seam configuration. West Virginia alone has over 50 minable seams (1).³ In the past, mining sequence was based primarily on availability and economics with little regard to the effects mining would have on other coalbeds both above and below the one being mined. This has strong implications for resource conservation, especially if these practices

continue. Problems that result from strata interactions could render these resources unminable unless methodologies and techniques are developed that allow for economical extraction.

To verify these interaction mechanisms in the field and their influence on mine ground stability, the Bureau conducted geological and geotechnical investigations. The purpose of these studies is to develop a better understanding of subsidence and pillar load transfer and its effects on current workings. Eventually, this knowledge will lead to improvements in mine planning and development.

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

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PILLAR LOAD TRANSFER

Pillar load transfer, as the name implies, involves the transfer of load pillars in overlying workings to pillars in underlying workings. This interaction occurs particularly when coalbeds are in close proximity, less than 110 ft apart (2-4), and either isolated, remant pillars (barriers) or many strong, competent pillars are present in the upper workings. This condition may serve to concentrate stresses in the innerburden, causing ground instability in the lower workings. The mechanics of load transfer and distribution from overlying operations have been analyzed extensively by other researchers through the use of mathematical and photoelastic models (3-4).

Two theories have been developed to explain ground disturbances due to load transfer from adjacent workings: pressure bulb theory and arching theory. Pressure bulb theory (3-4) assumes that the pillar is the major structural element in the transfer of load. It is useful in analyzing pillar load transfer when a "passive" interaction occurs. This condition is met when pillars are columnized and lower seam pillars are sufficiently large to prevent them from yielding. Arching theory (3-4) assumes that the mine opening is the major structural element in the transfer of load. Load transfer is the result of the pressure arch that forms around the mine opening upon excavation. This theory is useful in analyzing pillar load transfer when a "reactive" interaction occurs. This condition is met when lower seam pillars yield and redistribute their load to larger barriers or abutments.

CASE STUDY

Mine Location and Geology

The study mines are located in Raleigh County, WV, as shown in figure 1. The company is operating in two superimposed coalbeds. The upper mine is located in the Peerless Coalbed, which is approximately 72 in thick. The lower mine is located in the No. 2 Gas Coalbed, which is approximately 48 in thick. The mines

are separated by approximately 40 to 45 ft of innerburden.

The overburden consists predominantly of sandstone with interbedded shale units of varying thickness. The innerburden is a predominant sandstone with some interbedded shale units.

Depth

Depth in relation to all room-and-pillar mining operations is critical as overburden increases (3-4). The overburden above the lower mine at the study site is approximately 1,000 ft, the innerburden is approximately 40 to 45 ft thick, and approximately 960 ft of cover is located above the study site in the upper mine. Overburden depth changes dramatically in the area and reaches a topographic high over the study site.

Innerburden Thickness and Physical Characteristics

Interval thickness between two coalbeds is also a critical variable. Figure 2 is an innerburden isopach map constructed from available corehole information. As shown on the figure, approximately 43 ft separates the upper and lower mines at the study site. Core logs show that sandstone comprises 77 pct of the innerburden.

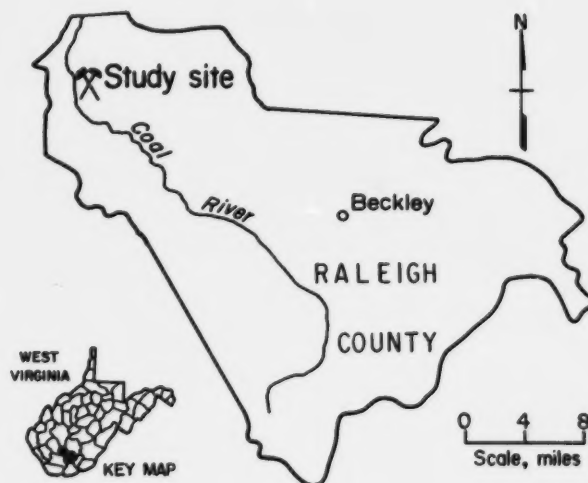


FIGURE 1.—Location of study mines.

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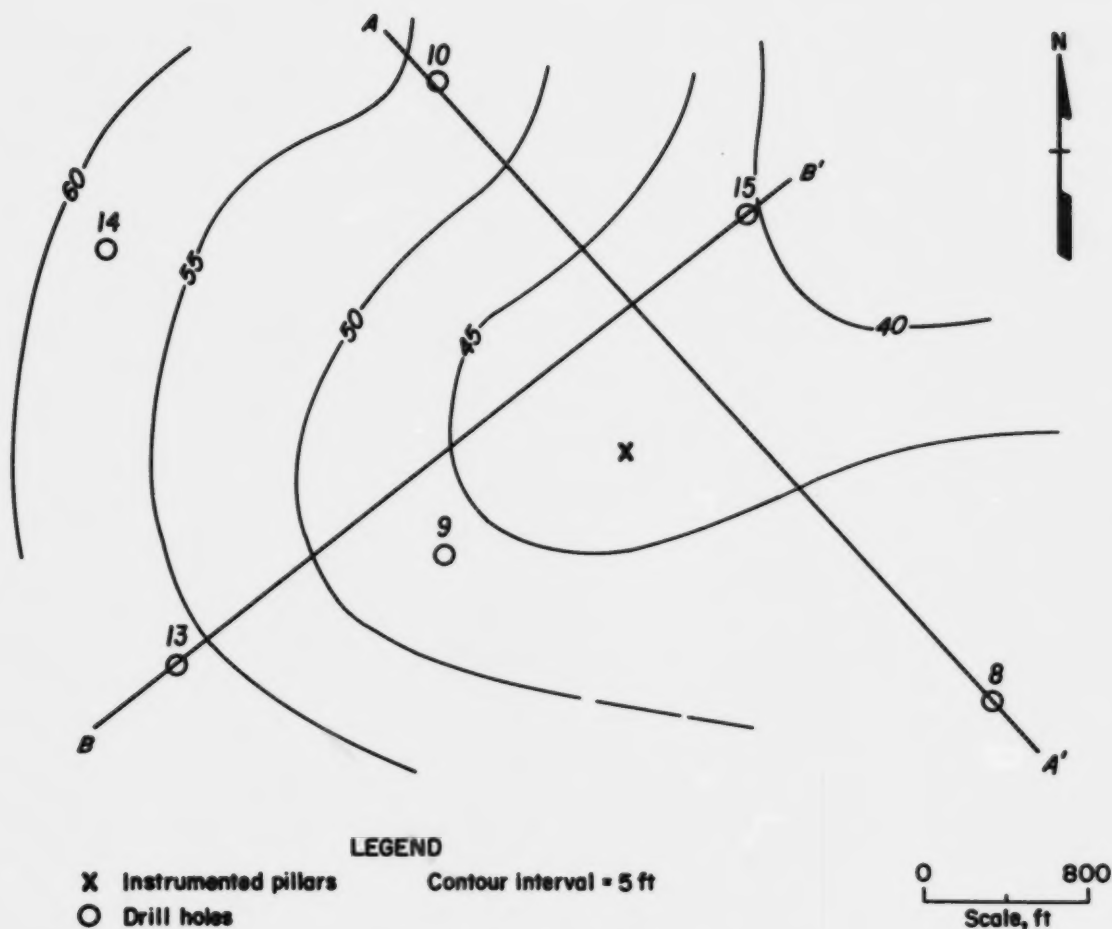


FIGURE 2.—Innerburden isopach map.

According to an equation developed by Haycocks and Karmis (3-4), approximately 78 ft would be the innerburden spacing above which no interaction damage may result from room-and-pillar mining. The actual innerburden with respect to the study site is approximately 40 ft, considerably less than the calculated minimum value. However, it should be noted that the equation is independent of pillar and entry design and is derived from a rather limited data set and therefore may not represent all stable and/or unstable mining conditions.

Results of In-Mine Mapping

A geologic investigation was performed within both the upper and lower mines. The areas of floor heave in both mines

are underlain by a rooted, fine-grained, well-cemented, hard sandstone and/or shale, and the roof consists of either a hard, banded siltstone or sandstone. Roof structure was generally uniform except for some slight undulations in the shale-sandstone contact. The immediate roof rock in the heave zones of the lower mine appears to be sandstone. In areas where a shale roof was observed, no floor heave was detected. The heave zones in the upper mine could not be correlated to roof rock lithology since sandstone is present throughout. Figures 3 and 4 represent the results of in-mine mapping. Figure 3 shows the location of floor heave around the study site in the upper mine. Note that the mine roof is chiefly sandstone near the study area and the floor is comprised of hard, fine-grained,

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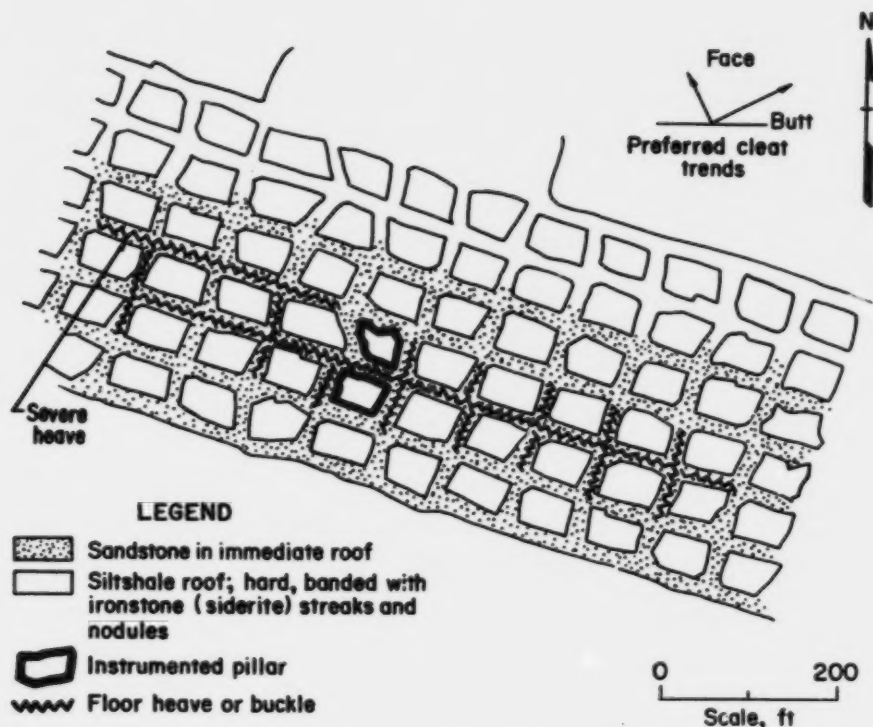


FIGURE 3.—Results of in-mine mapping of upper mine.

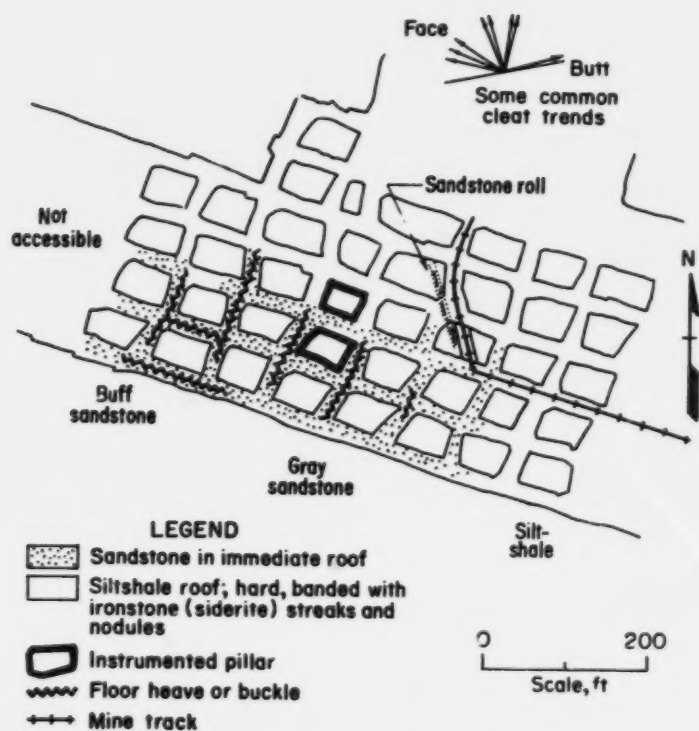


FIGURE 4.—Results of in-mine mapping of lower mine.

rooted sandstone. Figure 5 was constructed from core logs and displays floor lithology in the area of the study site. Note the lateral changes in the immediate floor lithology. The immediate floor is comprised of both a high-modulus material, a sandstone, and a low-modulus material, a shale.

In the heaving experienced near the study site, there was a sandstone floor. Inby and outby the study site, major floor heaving was experienced where a shale floor was present. Floor heaving experienced in these areas was observed to be humplike structures characteristic of a low-modulus material. A buckling type of failure was also observed where a sandstone floor was present. It may be assumed that major concentrations of load can be transferred from a shale (low-modulus material) to a sandstone (high-modulus material), which may result in movement within the sandstone floor. Figure 4 represents the results of the

in-mine mapping performed within the lower mine. The immediate floor located within the heave zones is also comprised of sandstone. Heave zones located in shale floor were found and also demonstrated humplike structures. The transfer of load from a shale to a sandstone unit could also be related to the ground conditions located within the lower mine.

Mining Engineering Design Parameters

Seam Sequencing

The upper mine area was driven in June 1980, and the same section located in the lower mine was driven during December 1982. Major floor heaving and excessive pillar loading were observed in October 1984 within the lower mine. Approximately 3 to 4 months later the upper mine experienced excessive entry convergence and pillar loading. Affected areas are shown in figures 3 and 4.

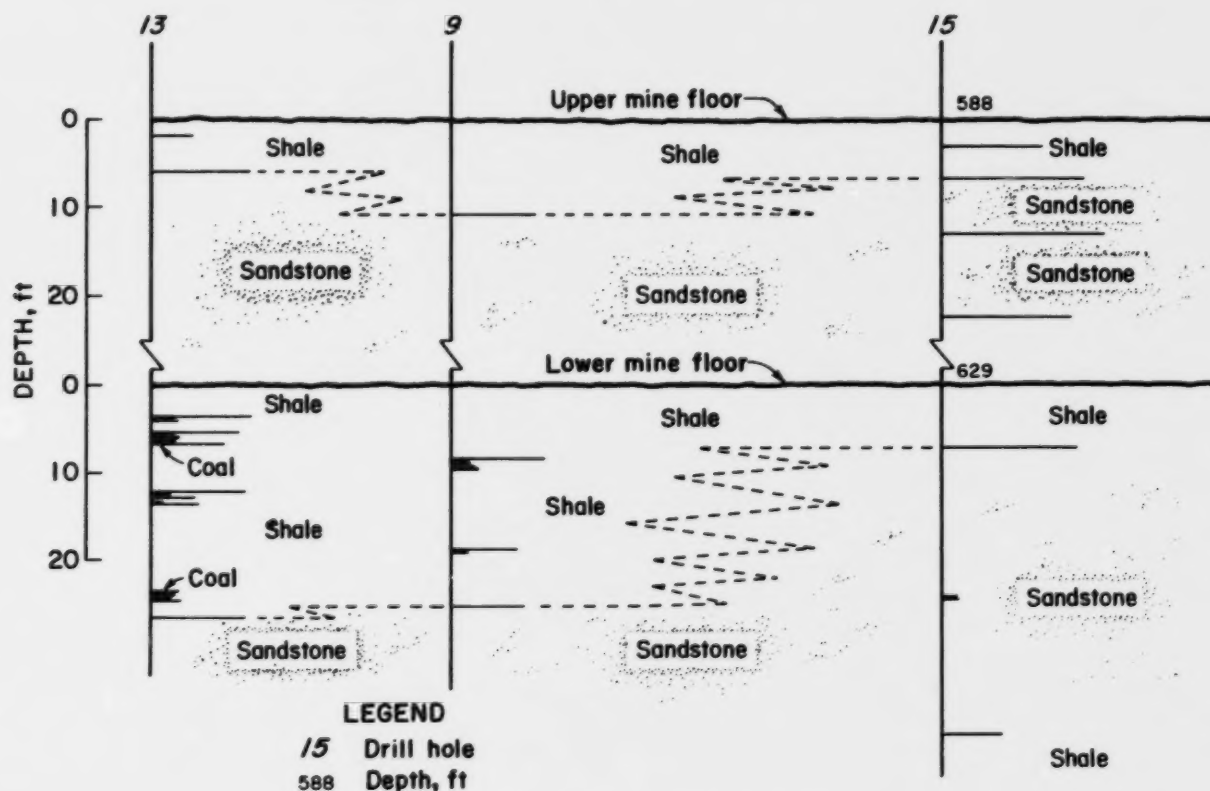


FIGURE 5.—Floor lithology in study site area.

Superpositioning of Pillars

The use of columnized pillars is standard practice in multiple-seam mine design. Columnization of the pillars lessens the effect of interaction that may be transferred from overlying workings. Figure 6 represents the superpositioning of pillars at the study site. Although very difficult to achieve, this practice requires alignment of pillars of similar size for both seams. Both the upper and lower mines were driven with pillars on 70-ft centers. Note from figure 6 that pillars and entries are nearly, but not totally, superpositioned.

Instrumentation and Results

Instrumentation

Instrumentation installed in both mines included borehole platened flatjacks (BPF) (5) and removable convergence

stations. The pillars selected for the study (fig. 6) are nearly superpositioned and have equivalent dimensions.

Four BPF's and 12 convergence stations were installed in the upper mine (fig. 7), and 5 BPF's and 12 convergence stations were installed in the lower mine (fig. 8).

The estimated setting pressures for the BPF's were calculated using the tributary area method.⁴ At the time of installation, depth and extraction ratio were estimated at 720 ft and 0.35 respectively. This gives setting pressures of 1,200 psig for the upper mine and 1,300 psig for the lower mine. It was later

$$^4 \text{Estimated pressure, psi} = 1.1 \text{ psi/ft} \\ (d) \frac{1}{(1-R)}$$

where d = depth, ft,

and R = extraction ratio.

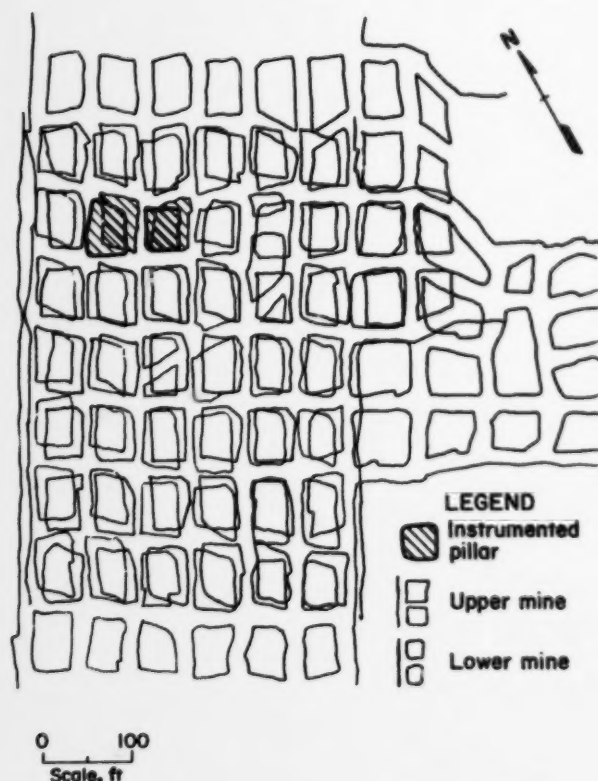


FIGURE 6.—Superpositioning of instrumented pillar.

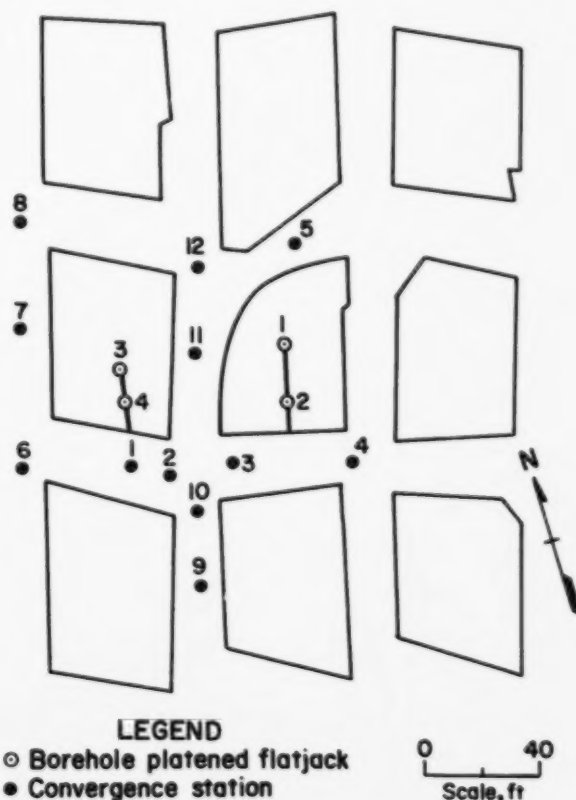


FIGURE 7.—Instrument location in upper mine.

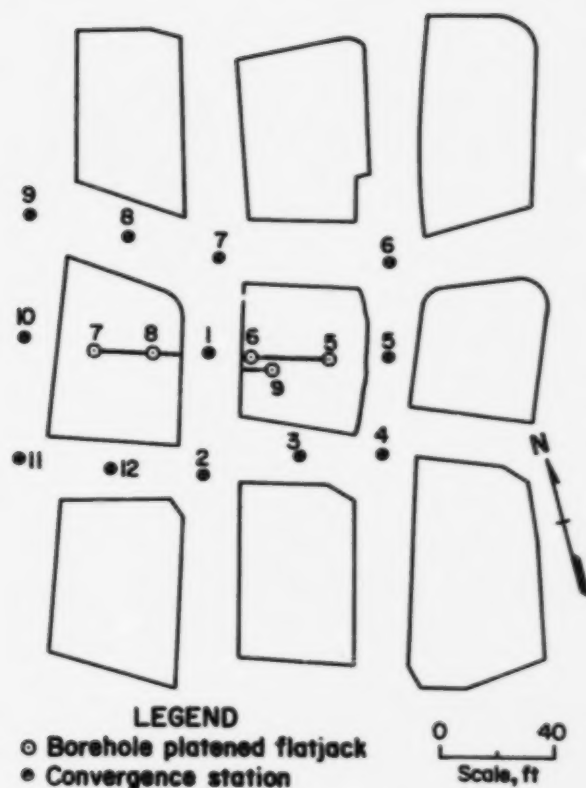


FIGURE 8.—Instrument location in lower mine.

determined that depth and extraction ratio were underestimated and were actually 960 ft and 0.50. As a result, setting pressures for the upper and lower mines should have been 2,100 psig and 2,200 psig respectively. Although the original setting pressures were low, these pressures do not directly affect the recorded results. Any increase in pillar pressure

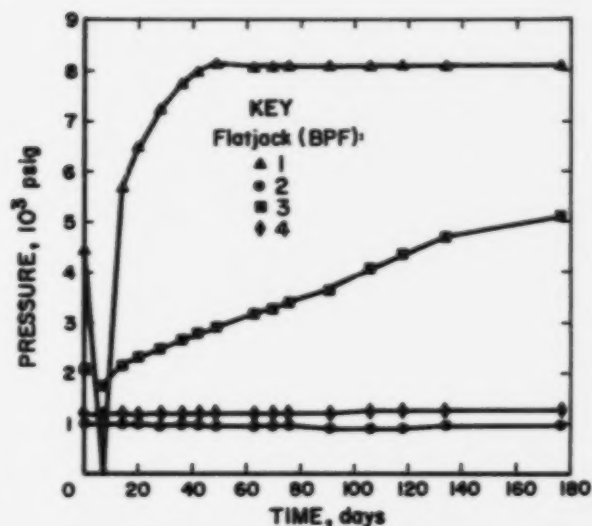


FIGURE 9.—Pressure increase versus time in upper mine.

above 2,100 psig and 2,200 psig would be a result of relative increases in pillar pressure.

Results

The monitoring of the instrumentation continued for a total of 177 days. The instruments were monitored at least once a week. Tables 1 and 2 provide results at 49 days (approximately 25 pct of the study period), 92 days (approximately 50 pct of the study period), and 177 days. All increases in pillar pressure for the upper mine were from BPF's installed within the core of the instrumented pillars. The BPF's located at 10-ft depths showed no increase in pressure. Figure 9 shows pressure increases versus time for

TABLE 1. — Flatjack (BPF) pressure during 177-day monitoring period, pounds per square inch

BPF	Initial (installation date)	Day 49 (25 pct of total period)	Day 92 (50 pct)	Day 177 (final)
UPPER MINE				
1.....	1,100	8,100	8,100	8,100
2.....	1,225	950	900	950
3.....	1,200	3,175	4,050	5,100
4.....	1,275	1,200	1,250	1,250
LOWER MINE				
5.....	1,300	1,090	1,050	1,050
6.....	1,000	880	850	850
7.....	1,300	1,050	1,000	1,050
8.....	1,300	900	900	900
10.....	1,200	1,000	1,000	950

TABLE 2. - Results of convergence monitoring in upper mine, inches

Station	Day 49	Day 92	Day 177 (final)	Station	Day 49	Day 92	Day 177 (final)
1.....	1.40	1.95	3.54	7.....	1.15	1.85	3.00
2.....	.84	1.25	2.13	8.....	.81	1.26	2.08
3.....	ND	ND	ND	9.....	2.23	3.26	5.00
4.....	.60	1.29	¹ 1.80	10.....	.94	1.31	2.12
5.....	1.24	1.88	2.71	11.....	.80	1.13	1.92
6.....	1.32	2.03	3.37	12.....	1.40	1.45	2.24

ND No data because station was destroyed.

¹Monitoring discontinued on day 134 owing to bad roof conditions.

all BPF's installed in the upper mine. No increases in pillar pressure were recorded from BPF's installed in the lower mine. This was due to height restrictions which limited instrument installation to more stable mine areas.

Maximum convergence of 5 in occurred at station 9, which is located in the track entry within the upper mine. Roof-to-floor convergence within the upper mine increased very rapidly, averaging 0.5 in

of closure every month. Figures 10 and 11 show total convergence within the upper mine after 49 days (25 pct of monitoring period) and 177 days (100 pct of monitoring period). Note the trend of movement with respect to figure 11. Movement occurred outby and in a southwest direction in relation to the instrument array. Figure 12 displays total convergence versus time for the four convergence stations located along this

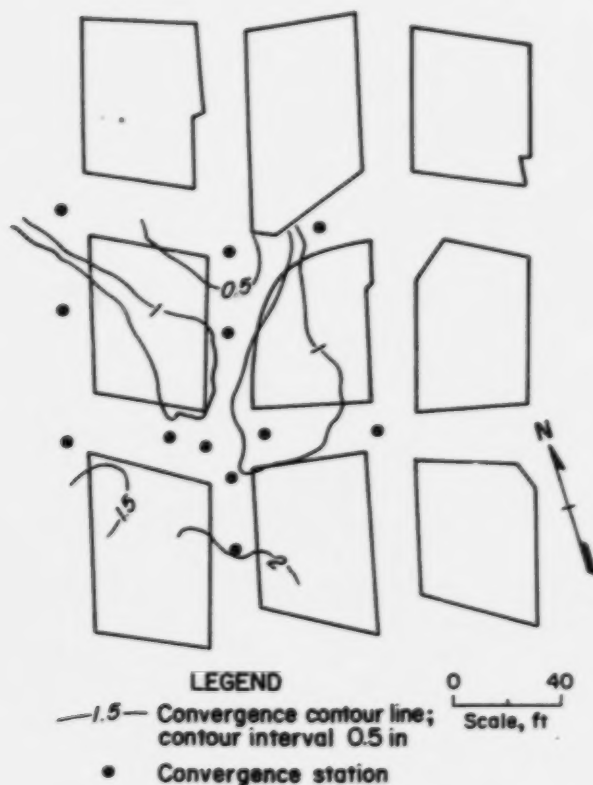


FIGURE 10.—Upper mine convergence contours after 49 days of monitoring.

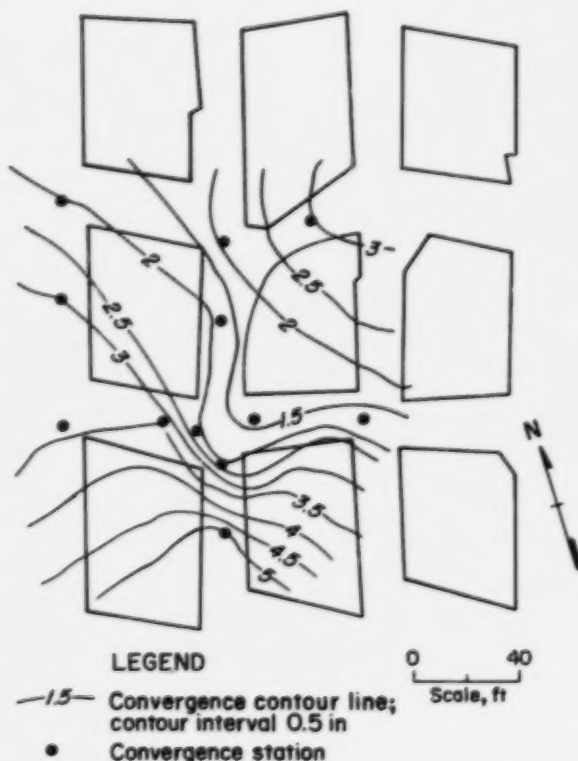


FIGURE 11.—Upper mine convergence contours after 177 days of monitoring.

trend of major roof-to-floor movement. Roof-to-floor convergence monitored in the lower mine was limited. The most movement that occurred was 0.13 in at station 11. Figure 13 represents total convergence (117 days) monitored within the lower mine.

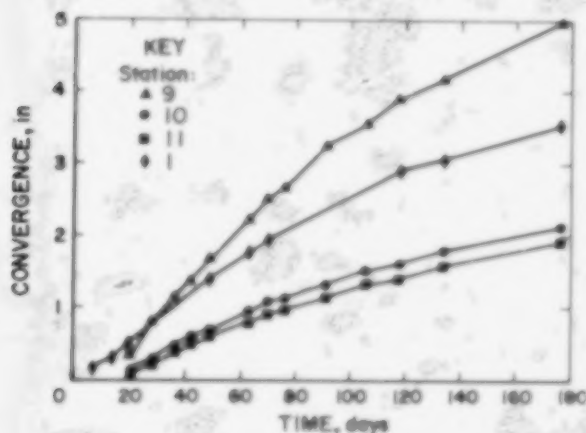


FIGURE 12.—Total convergence versus time in upper mine.

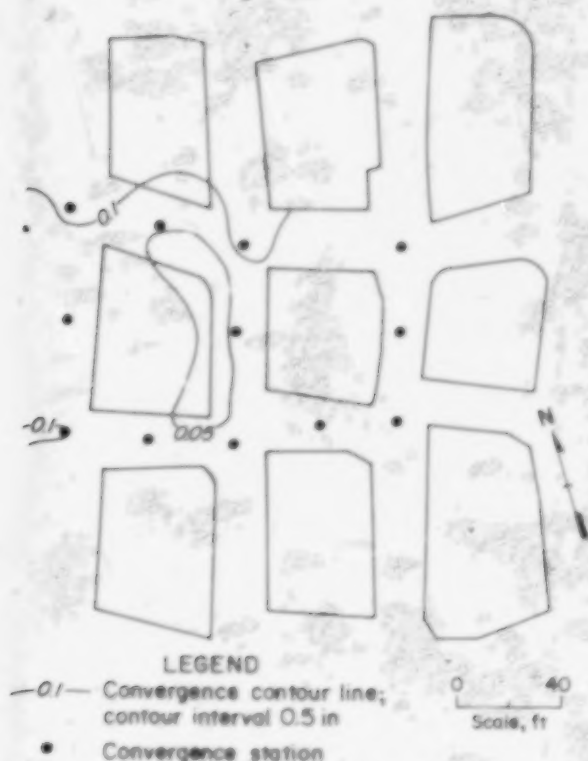


FIGURE 13.—Lower mine convergence contours after 177 days of monitoring.

CONCLUSIONS ON PILLAR LOAD TRANSFER

Based on the information received and collected throughout the study, the following conclusions can be made:

1. Overburden depth above the study site was approximately 1,000 ft. Prior research and other case studies (2-4) have shown that excessive overburden depths can lead to unstable ground conditions.

2. Innerburden thickness in the study area was approximately 40 to 45 ft, less than one pillar width. Prior research has shown (3-4) that workings in close proximity, less than two pillar widths, may create ground control problems in, above, and below workings.

3. Prior research has shown (3-4) that innerburden material comprised mostly of sandstone dampens the effects of pillar load transfer. Sandstone at the study area comprises 77 pct of the innerburden. According to Haycocks (3) this percentage requires 78 ft of innerburden for stable conditions.

4. Heaving was experienced in both sandstone and shale floor units. The shale floor was a low-modulus material, resulting in humplike floor heaving. Whereas the sandstone floor was a high-modulus material, resulting in a buckling type of floor heave.

5. Average convergence in the upper mine entries was 2.50 in, compared with 0.04 in. in the lower mine entries. This difference is due to height restrictions in the lower mine, which limited instrument installation to a more stable mine area.

6. To minimize interaction effects, optimum seam sequencing would be to mine the upper seam first to total extraction and then continue downward. In this case, the upper seam pillars were developed first, and the lower seam pillars were developed approximately 2 yr later. Pillar columnization was practiced, but mine overlays show that pillars and entries were not totally superpositioned.

It is difficult to determine whether columnized pillars cause a "passive" interaction through pressure bulb interference or if a "reactive" interaction

occurred owing to arching effects. BPF pressure readings in the upper mine show a core loading that is characteristic of a stiff pillar approaching failure. Similar loading characteristics were not observed in lower seam pillars, mainly because height restrictions limited BPF installation to more stable areas. But assuming lower seam pillars also exhibit stiff pillar characteristics, the arching concept could be applicable in this situation. Photoelastic and mathematical models of multiple openings in close proximity, less than two pillar widths, have shown that pressure arch interaction can create zones of excessive pressure in the innerburden (3-4). This is especially the case for strong, competent pillars or stiff pillars that do not yield

readily, allowing independent arches to form from pillar to pillar. When high abutment pressures associated with arch interaction exceed the in situ strength of the rock, ground failure results. In this case, a soft floor stratum is the weakest member in the mine structure. Strong competent pillars punching into the floor cause considerable floor heave, which is then followed by eventual pillar failure or yielding. When this occurs, their load is transferred to neighboring pillars, forming a secondary arch. This cycle of failure, load transfer, and arch formation continues until sufficient support is encountered to stabilize the load transfer process, such as barrier or abutment pillars.

SUBSIDENCE

Strata interactions due to subsidence result when an underlying coalbed is extracted first. Undermining subjects the superjacent strata to a mining-induced stress field (6). The distribution of this stress within the strata is a function of the subsidence process and is most damaging to overlying coalbeds after the critical to supercritical subsidence phase has been reached (4, 6-7). Depending upon the uniformity of lower coalbed extraction, there exists a relatively distressed zone toward the middle of the subsidence area. Most ground disturbances in overlying coalbeds occur toward

the boundaries of the subsidence trough. Within the trough, strata flexure creates zones of tensile and compressive stress, as shown in figure 14 (17). The extent of this zone is defined by the angle of draw, which is dependent upon the geologic and physical characteristics of the strata. As mining develops through this trough, these stresses have a severe effect on entry stability, particularly on the integrity of the roof.

Other types of failure in upper coalbeds that are attributed to undermining include interseam shearing and the effects of arching. In interseam shearing, highly inclined shear or tensile-shear failures develop and result in displacement of the coalbed into lower excavations (4, 7). Recent studies (4, 7) indicate that the elastic modulus of the superincumbent strata is a major factor influencing this type of failure. Studies have demonstrated that a high-modulus strata such as sandstone is more prone to shear failure. Arching is actually a subcritical phase of subsidence, and its effects on upper coalbeds are dependent upon the opening width-to-depth ratio and the height of the resulting pressure arch. Arching effects can produce a zone of high compressive stress that may cause pillar and roof control problems.

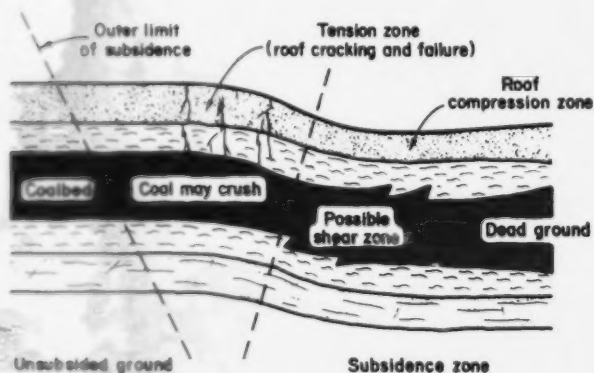


FIGURE 14.—Strata flexure in upper coalbed due to subsidence. Adapted from Haycocks, Karmis, and Topuz (7).

CASE STUDY

Mine Location and Geology

The study mine is located in Greene County, PA, as shown in figure 15 and is operating in areas of the Sewickley Coalbed that were subsided by undermining of the Pittsburgh Coalbed. The overburden above the Sewickley Coalbed ranges from 425 to 580 ft and consists predominantly of interbedded shale with a competent sandstone unit that varies in thickness. The innerburden ranges from 90 to 100 ft and consists of interbedded shale and limestone.

The average height of the Sewickley Coalbed is 5 ft. The immediate roof is composed of a highly jointed dark sandy shale that ranges from 10 to 15 ft thick, overlain by a competent limy shale. The immediate floor is composed of a 3-ft-thick, dark, limy shale underlain by a competent limestone unit. An underground geologic investigation found no geologic anomalies (clay veins, discontinuities, etc.) in the study area. Additional site-specific information is given in table 3.

Instrumentation and Results

1 Left Panel

1 Left was a short 400-ft panel that was developed through a subsided area of the coalbed created by pillar retreat activities in the lower mine. The boundary

TABLE 3. - Site-specific information,¹
Greene County, PA, mine

	Upper mine ²	Lower mine ³
Av mining height...in..	60-64	72
Pillar centers.....ft..	100×100	100×100
Av entry width.....ft..	18-20	20
Extraction, pct:		
Mains.....	36	36
Panels.....	100	100

¹Room-and-pillar continuous mining at both levels.

²Sewickley Coalbed.

³Pittsburgh Coalbed.

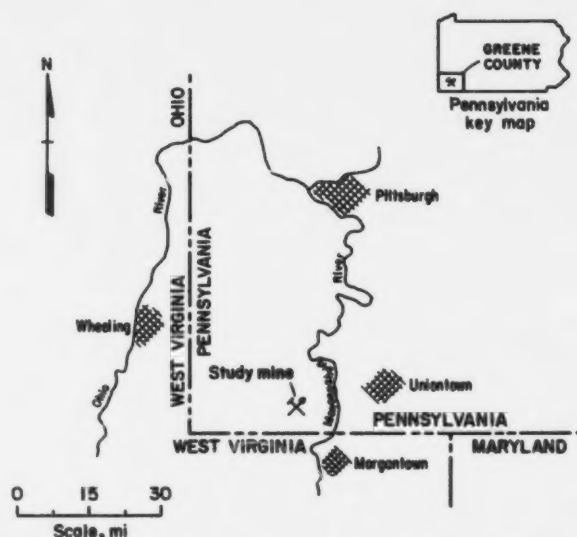


FIGURE 15.—Study mine location, Greene County, PA, mine.

of this retreat mining in the Pittsburgh Coalbed is shown as the gob line in figure 16. The 1 Left panel started in subsidence (over gob) and developed across the gob line and onto pillars located in the lower mine. It was developed and retreated in less than 50 days. Several years earlier, the 1 East panel crossed the gob line (from over support pillars and onto gob in the lower mine); it experienced two roof falls as shown in figure 16 but encountered no displacements in the coalbed. These ground conditions indicated that the strata directly superjacent to the gob line were flexed due to subsidence, and mine personnel were anticipating these same conditions in the 1 Left panel. To predict the location of the subsidence trough within the coalbed is a complex problem. Although current theories and models are based on surface subsidence, their application for predicting in-mine subsidence could prove useful. Based on a model recently developed by the Bureau (8-9), the edges of the subsidence trough at the Sewickley Coalbed level were calculated to be approximately 102 ft inby to 300 ft outby the gob line of the lower mine for this case. The edges of the predicted trough and pillar arrangements in 1 Left before retreat mining are shown in figure 17.

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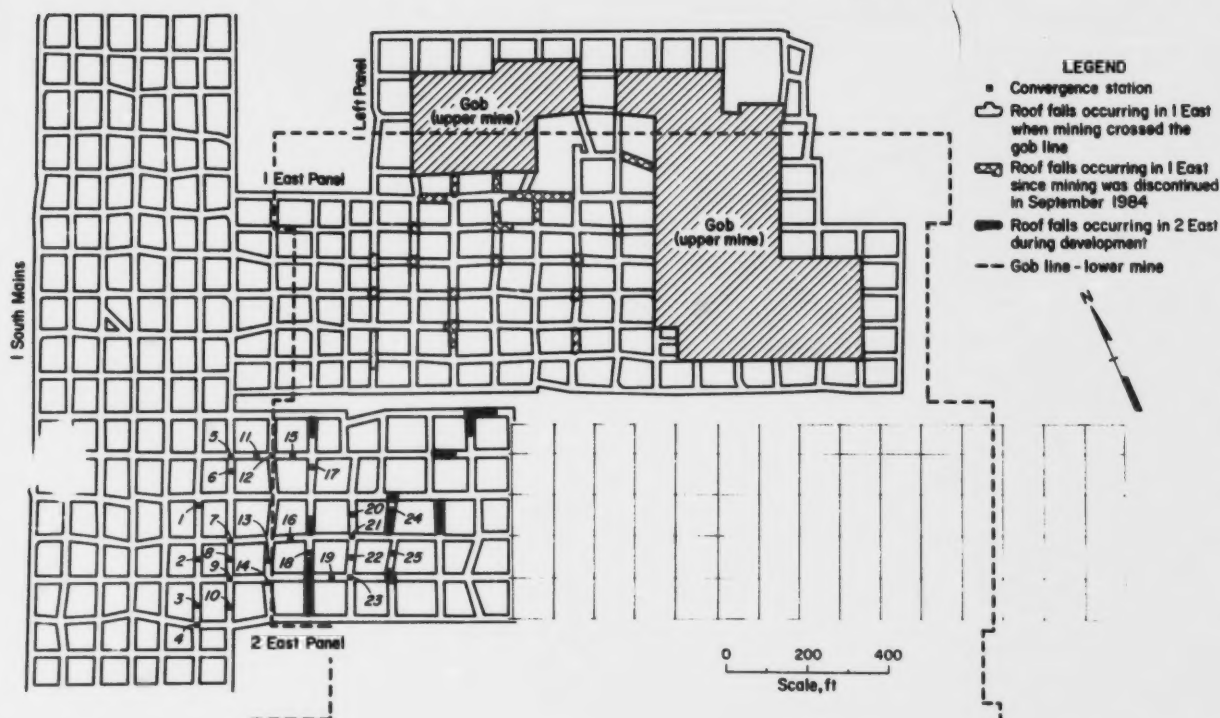


FIGURE 16.—Location of study areas 1 Left and 2 East showing gobline in lower mine (Pittsburgh Coalbed) and roof fall activity in upper mine (Sewickley Coalbed).

To determine if the trough affected pillar loading and stability, four BPF's (12) were installed during development to measure vertical changes in pillar pressure. Setting pressure was approximately 1,000 psig for each BPF⁵ as overburden was 580 ft with an extraction ratio of 0.36.

Figure 17 shows BPF locations in selected pillars. During development the instruments recorded no increases in pillar pressure. The major ground problem experienced within the trough during development was the occurrence of a large roof fall, 10 to 12 ft high, over gob as shown in figure 17. This fall started in a crosscut at the face and eventually propagated into the belt entry, causing considerable production delays.

BPF pressure increased, as anticipated, when retreat mining approached the instrumented pillars. Figure 18 shows the pressure changes recorded for the four BPF's during development and retreat

mining. Pressure increases ranged from 400 to 700 psig but were not considered significant because they did not render a pillar or adjacent pillar unminable during retreat. Observations of roof behavior at the face showed the roof strata to fall tight against the pillar line, causing no excessive loading in adjacent pillars from roof cantilevering. The 1 Left panel was completely retreated, but ground water inflow and accumulation slowed retreat operations in the 1 East panel. Eventually, retreat mining was discontinued completely in the 1 East panel due to this condition.

2 East Panel

2 East was a long 2,200-ft panel developed off the 1 South Mains. It started from over support pillars located in the lower mine and developed across the gob line into subsided ground as shown in figure 16. To better understand immediate ground movement as entries were developed through the subsidence trough,

⁵See footnote 4 (p. 32).

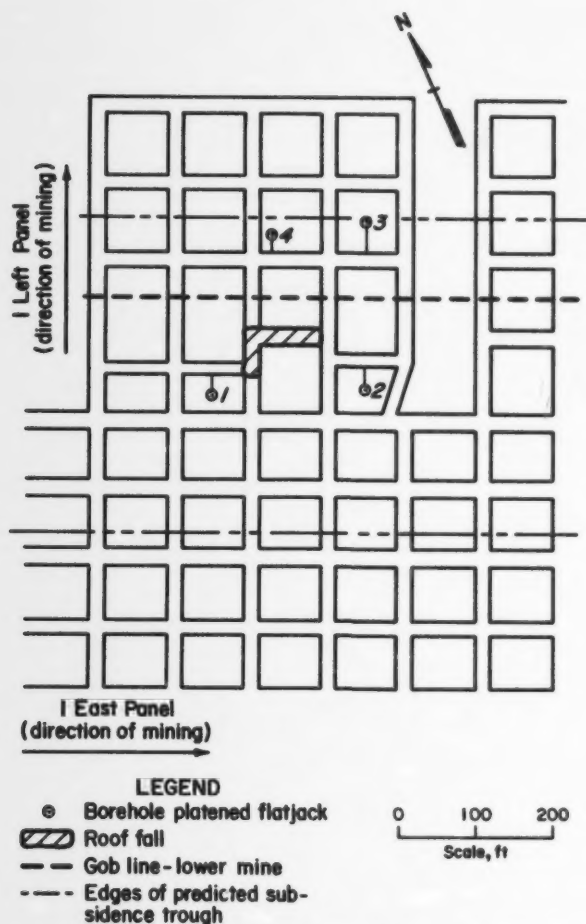


FIGURE 17.—Pillar arrangements and BPF locations in 1 Left before retreat mining.

roof-to-floor convergence measurements were used in the 2 East panel. A total of 25 convergence stations were installed within the trough area, 14 in entries developed over support pillars and 11 in entries developed over the gob of the lower mine. As shown in figure 16, large roof falls 10 to 12 ft high occurred over gob during pillar development, but with greater frequency than in other gob line crossings. Convergence measurements taken over 143 days showed that the average total convergence measured in entries developed over the gob was over four times the amount measured in entries developed over support pillars. Figure 19 shows cumulative convergence for stations 1, 8, and 13 installed in entries

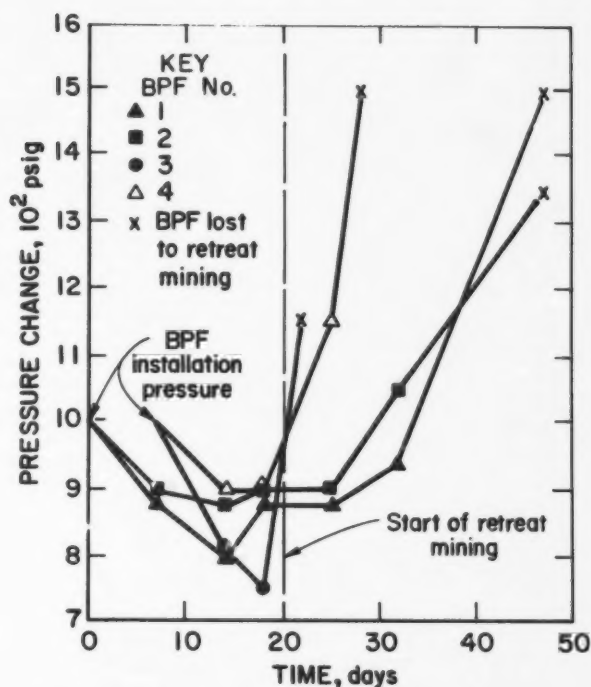


FIGURE 18.—Pressure changes recorded for BPF locations in 1 Left before development and retreat mining.

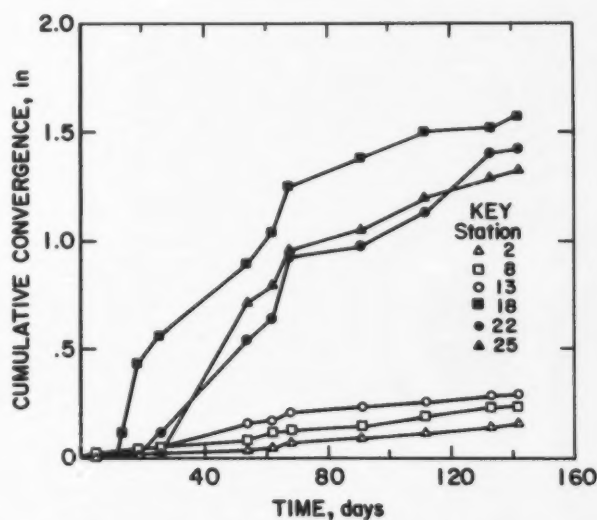


FIGURE 19.—Cumulative convergence for stations 2, 8, 13, 18, 22, and 25 over 143 days.

developed over support pillars and stations 18, 22, and 25 installed in entries developed over gob. This graph shows that the rate of convergence is nearly

uniform on both sides of the gob line but increases dramatically once the gob line is crossed. Table 4 lists cumulative convergence measured over 143 days for all stations.

CONCLUSIONS ON SUBSIDENCE

Throughout the subsidence area, the coalbed showed no vertical displacements that would indicate an interseam shearing, and no recurring problems where a result of strata flexure. Several observations support this:

First, as mining development crossed the gob line in the 2 East panel, roof-to-floor convergence increased, as did

TABLE 4. - Cumulative convergence measured over 143 days for all stations

Station	Cumulative convergence, in
Entries developed over support pillars:	
1.....	0.11
2.....	.15
3.....	.16
4.....	.13
5.....	.19
6.....	.12
7.....	.15
8.....	.24
9.....	1.39
10.....	.27
11.....	.16
12.....	.17
13.....	.31
14.....	.95
Entries developed over gob:	
15.....	.33
16.....	.96
17.....	1.11
18.....	1.56
19.....	1.50
20.....	2.71
21.....	1.46
22.....	1.38
23.....	1.38
24.....	.73
25.....	1.33

the incidence of roof falls. In figure 20, graph 1 is the predicted subsidence profile (8-9) for this case, graph 2 shows cumulative convergence measured over 143 days for selected stations 2, 8, 13, 18, 22, and 25 and their locations with respect to the gob line, graph 3 depicts the cumulative length of roof falls with respect to the gob line, and graph 4 is a plot of the overburden above the 2 East panel. Graph 1 shows that the subsidence trough begins 102 ft outby the gob line and subsidence reaches a maximum at 300 ft inby the gob line. Graphs 2 and 3, which depict ground movements, correlate well with this predicted subsidence profile as roof-to-floor convergence and frequency of roof falls both increase dramatically as the gob line is crossed. Graph 4 shows there were no dramatic fluctuations in overburden above the panel. Second, all roof falls in the study area occurred in entries developed

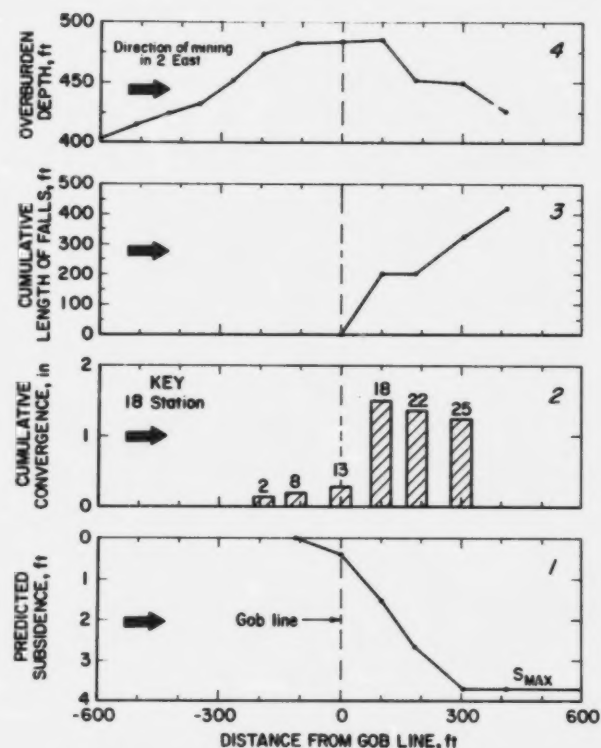


FIGURE 20.—All graphs in relation to gobline of the lower mine. 1, Predictive subsidence; 2, cumulative convergence; 3, cumulative length of roof falls; 4, overburden depth.

over gob; no roof falls occurred in entries developed over support pillars. In addition, the roof fall activity continued in the 1 East panel well after retreat mining operations were discontinued in September 1984 (fig. 16).

Finally, most roof falls displayed a similar type of roof failure, usually along a natural roof joint. As shown in figure 21, these joint surfaces were smooth, having no cohesive properties. This natural jointing system was present throughout the study area, yet as mentioned earlier, all roof falls occurred in entries developed over gob. Presuming that the roof is in flexure, these joint surfaces would provide natural planes for tension failure. A major set of roof joints was oriented approximately N 55° W, subparallel to the direction of mining. This could further explain the high frequency of roof falls in intersections and entries parallel with the face.



FIGURE 21.—Failure along a roof joint.

The increased incidence of roof falls within the subsidence trough required the installation of additional roof support. Roof stability in entries developed over support pillars was readily maintained with 6-ft conventional bolts on 4-ft centers. Cribs and posts were installed as dictated by the general roof support plan. Entries developed over the gob required more comprehensive roof support consisting of 5-in by 7-in by 16-ft timbers bolted on 2-ft centers. In addition, many intersections along the track and belt entries were supported with 6-in steel I-beams set on posts or cribbing. These support requirements, combined with downtime to clean and resupport fall areas, lowered production considerably. Figure 22 shows the average tonnage per shift as the 2 East panel mined through this subsidence trough versus tonnage for a similar panel, 2 North, located in the same mine but not affected by subsidence. Note that the high production values for the 2 East panel are less than the low values for the 2 North panel. As mining progressed farther into the subsided zone, roof conditions improved slightly and production increased, but this supplemental support was still required.

Based on information collected at this particular study site, the following conclusions can be made:

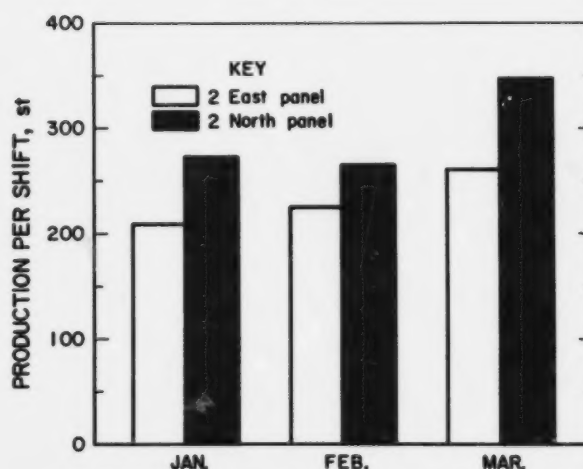


FIGURE 22.—Production figures, January-March 1985 - 2 East versus 2 North.

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1. Subsidence in the Sewickley Coalbed, caused by undermining of the Pittsburgh Coalbed, resulted in lower production and increased roof support costs during development and retreat mining.

2. Entry convergence and roof fall activity increased dramatically as development crossed the gob line and proceeded into subsided ground. Falls occur predominantly in highly jointed zones in the roof strata. The direction of mining in relation to the direction of roof joints may contribute to roof fall activity.

3. Strata flexure has little effect on pillar stability when percent extraction is kept low, leaving sufficient coal in place. In this case, pillars were 80- by 80-ft with 36-pct extraction.

4. The predicted location of the subsidence trough correlated well with recorded ground movements and observed ground behavior. The application of this model for predicting in-mine subsidence will be studied further at this and other sites.

DISCUSSION

Interactions between adjacent workings that are a result of pillar load transfer are difficult to predict, for the complex nature of the mechanism is not yet fully understood. Once the mechanism responsible for this interaction is better defined through further laboratory and field research, problem areas may be anticipated and possibly eliminated. Initial research indicates that the pressure arch theory is a feasible concept in understanding the interaction mechanism but requires further study. The literature reports in many cases that pillar superpositioning and a mining sequence of upper coalbed first can increase the stability of lower workings, but this is not always entirely effective. Barriers and abutment pillars are also beneficial in limiting interaction effects by providing immediate support to load transfer. Problems associated with subsidence of an overlying coalbed can be reasonably predicted through careful premining planning and analysis. This should include accurate maps of lower mine workings, a geologic evaluation of innerburden and overburden, and the mapping of roof joints and geologic features. This information together with other site specifics (age of workings, uniformity of pillar

extraction, mining height, etc.) should be used collectively to anticipate areas within the coalbed where problems may result or be magnified by undermining. Continued research into the development and application of a subsidence model will improve the accuracy of prediction.

Minimizing problems may require changing the normal mining or roof control plan. Some possible changes in the mining plan may include reducing percent extraction, decreasing roof span width, staggered pillar arrangements, and changing the heading of entries. The roof control plan may also require modification to contend with changing roof conditions. This may require different roof bolt types and lengths as well as increased supplemental supports.

Owing to economics, ownership, and availability, a mine operator may have little control over coalbed mining sequence. In those cases where underlying coalbeds are extracted first, subsidence in upper beds may result in lower production and increased roof support costs. Cooperative planning within the mining community can avoid this and lead to substantial improvements in resource conservation.

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INTEGRATED DESIGN FOR STABILITY IN MULTIPLE-SEAM MINING

By Chris Haycocks,¹ Wei Wu,² and Yingxin Zhou²

ABSTRACT

Research into ground control problems resulting from mining in a multiple-seam environment has been carried out using statistical analysis, numerical modeling, and body-loaded multilayer photoelastic analysis in conjunction with numerous case studies. Findings demonstrate situations under which the worst ground control conditions can be expected to occur for a specified geologic environment, depth, innerburden spacing, and mining geometry. Special emphasis has been placed on the effect of upper pillar load transfer on the lower seam in terms of

pillar and floor stability and roof control. Geologic and lithologic conditions in the affected seam are summarized by roof and floor index. The effects of joint orientation and continuity are also incorporated in the analysis. Upper seam effects due to undermining are given in terms of roof damage related to total vertical displacement with allowances for geologic variations. Design criteria for mining on a lower seam have been incorporated into a friendly interactive program for use on personal computers.

INTRODUCTION

The majority of Appalachian coal deposits lie in a multiple-seam environment. Historically, the decision to extract a particular seam has been based on ease of access, ownership, and economics rather than ground control considerations. As mining progresses and seams are mined out, it is increasingly common to find working and abandoned mines in close vertical proximity to each other. Depending on the sequence of extraction, such multiple-seam mining operations can be classified in terms of mining sequence as

undermining, overmining, or simultaneous mining. Contiguous operations can produce both positive and negative ground control situations in neighboring seams. These interaction effects are frequently reported, and their severity can range from improved opening stability to minor roof problems that may require no additional support to complete loss of a large section of coal reserves (1-4).³ As a consequence, interaction problems are of growing concern to the Appalachian underground coal industry.

FIELD STUDIES

Since many Appalachian coal mines have experienced interaction problems (1-2, 5), there are considerable case studies available for research and analysis. To date, over 130 separate examples of multiple-seam ground control problems have been collected to facilitate identification of major controlling factors and the magnitude and limits of interaction under a variety of conditions.

Analysis of field experiences reveals four major classes of interactive ground control mechanisms: pillar load transfer, innerburden shearing, arching effects, and upper seam subsidence (3, 6). Pillar load transfer mechanisms and innerburden shearing have been successfully used to explain interaction phenomena associated with undermining during overmining.

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

Factors that contribute to interaction may be classified into variables that are fixed by the geologic environment and those that depend on engineering design. These may be described⁴ as follows (6):

Fixed: Depth,* innerburden thickness,* innerburden physical characteristics,* stress field, seam thickness, coal characteristics, completed mining operations either above or below the seam to be mined, and age of workings.

Mining: Height, dimensions and geometry,* method,* and spatial location of entries.*

Sorting of data revealed that, considering all case studies, 46 pct reported relatively serious interaction problems. Among the mines experiencing interaction problems, simultaneous mining caused the most trouble and accounted for 48 pct

⁴Factors marked with an asterisk are regarded as being critical.

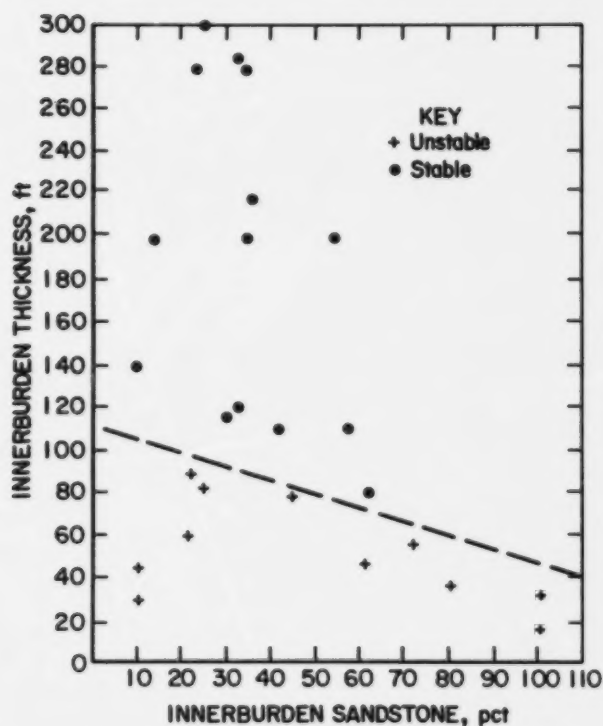


FIGURE 1.—Influence of percentage of hard rock in the innerburden on stability in the lower seam. Adapted from Haycocks (3).

of the problems; overmining and undermining accounted for 30 and 22 pct, respectively.

Initial analysis of field study data by plotting the data with respect to some related variables also demonstrated some simple but useful relationships among variables and some interesting trends (figs. 1-3).

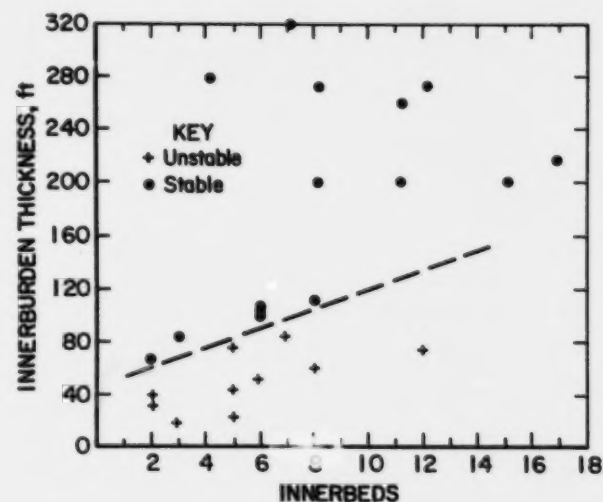


FIGURE 2.—Number of innerbeds versus innerburden thickness for stable and unstable lower seam conditions. Adapted from Haycocks (3).

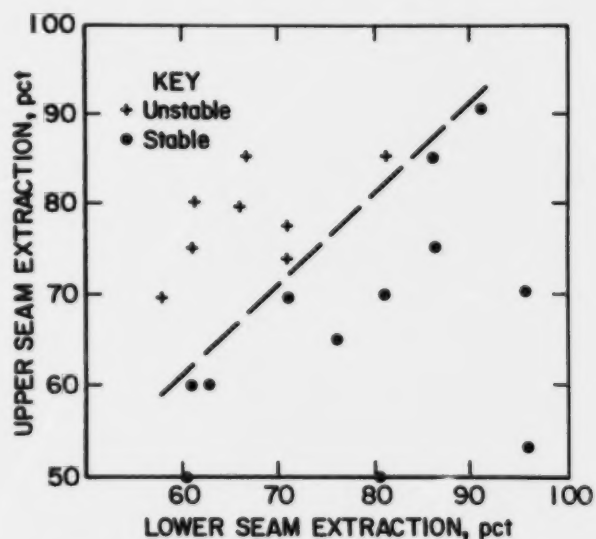


FIGURE 3.—Relationship between percent extraction in the upper and lower seams and lower seam stability. Adapted from Ehgartner (7).

Figure 1 establishes the innerburden limits for interaction problems due to pillar loading onto an underlying seam in terms of the percentage sandstone in the innerburden. Subsequent photoelastic analysis of this pillar load transfer phenomenon indicates that the degree of layering in the innerburden is as

important as the modulus, as shown in figure 2 (7). It was found that the relative extraction ratios in two adjacent seams are extremely important in determining whether or not interaction will be experienced. Higher upper seam extraction will increase the chance of unstable conditions in an adjacent seam (fig. 3).

STATISTICAL ANALYSIS

Since the field data collected are numerous and very complex, multivariate statistical procedures were used to gain further insight into the field data. Statistical analyses were also used to derive relationships between geological factors, which are generally difficult to incorporate into other analytic models (3, 8-10). For the proposed analysis, variable selection was primarily determined by former research experience and the availability of data. The appendix shows a total of 24 variables selected and used. Certain qualitative variables are used for classification purposes, and some of them were found very useful in separating the local problems due to in situ geological disturbances from interaction problems caused by mining excavations in adjacent seams.

Five multivariate analysis techniques were used in the analyses (11-12). Except for regression analysis, these procedures are similar in that they all try to transfer the original data set into a

new data set. By data transformation, some specific goals can be achieved which may lead to better understanding of the original problem (13-14).

RESULTS FROM PRINCIPAL COMPONENT ANALYSIS AND FACTOR ANALYSIS

A principal component analysis was performed on eight quantitative variables. By Kaiser's rule (eigenvalue > 1), or proportional criterion, three principal components were extracted that explain 71 pct of the total variance of the original data. Here, the cases with interaction can be better separated from those without interaction (fig. 4). Also, when the first component is large or the third component is small, the chance of interaction problems increases. The first principal component has relatively higher loadings on variables such as COVER, UST, USEP, and IBT while the third component has higher loadings on the variables IBSP and LSEP. (See appendix 1 for definition of variables.) This indicates that the interaction problem may be attributed to cover depth, upper seam thickness, extraction on both seams, and nature of innerburden. It is noted that innerburden (IBT) is negatively related to the first component. To decrease the score of the first component, and thus reduce the possibility of having interaction problems, the spacing between two seams must increase. The results from factor analysis showed a similar grouping of variables.

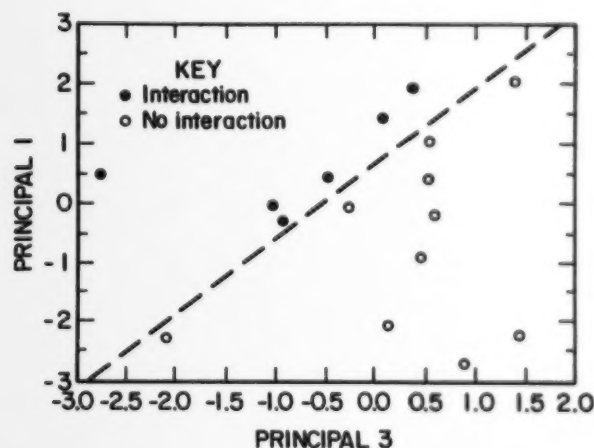


FIGURE 4.—Results from principal component analysis (overmining).

RESULTS FROM DISCRIMINANT ANALYSIS AND CLUSTER ANALYSIS

Discriminant analysis was applied to classify the multiple-seam mines into

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two groups, one with interaction problems and the other without, based on observed geological and mining conditions. Although no general way of classifying the observations for all mining methods was found, the classification of mines using overmining methods was successful. Figure 5 shows the scatter plot with two canonical scores of each observation as its coordinates. It can be seen that the group with interaction problems is well separated from the group without interaction effects. This implies that for overmining mines it is possible to predict interaction using the discriminant analysis. However, severity of interaction cannot be estimated by such a classification scheme.

When cluster analysis, like discriminant analysis, was used in studying all case study data, no natural groups or clusters could be identified. However, when this procedure was independently used for data from mines using overmining or undermining methods, it was possible to classify the observations according to location of interaction problems.

The location where the interaction occurred was originally classified into five categories: below or above remnant pillar or pillars (A or B), below or above the interface between gob area and large section of solid coal (C or D), or not as above (E). The resulted clustering indicates that most interaction problems can be grouped into four types of locations (A-D). These results suggest

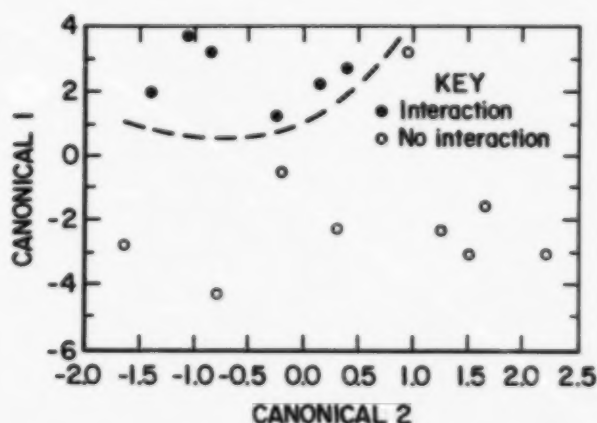


FIGURE 5.—Results from canonical discriminant analysis (overmining).

that the initial classification is correct and, in addition, the interaction problems occurring at the same location may have similar geological and mining conditions.

In summary, of all reported severe problems, 10 pct occurred either below or above a remnant pillar or pillars (A and B type locations) and 22 pct occurred below or above the interface between a gob area and large section of solid coal (C and D type locations). This demonstrates the possibility of interaction due to stress anomalies caused by isolated pillars left on the upper or lower seams.

RESULTS FROM REGRESSION ANALYSIS

When data were sorted into different mining methods, and innerburden thickness between adjacent seams was limited to less than or equal to 300 ft, good linear models were obtained for close-seam mines using stepwise regression procedures. For example, for mines with close seams and using overmining methods, the following regression equation can be derived:

$$\text{INTERP} = -2.13 + 0.32\text{USEP} - 0.38\text{LSEP} \\ + 0.12\text{IBSP} + 0.58\text{LST.}$$

This model can explain 78 pct of total variance (R -square = 0.7799) and has a C_p value of 2.76, which is the best compared to other possible models. The error mean square of this model is also very small. In a physical sense, the degree of damage due to interaction in overmining can be related to, according to sequence of importance, upper and lower seam extraction percentages, innerburden sandstone or strong rock percentage, and lower seam thickness.

Through similar procedures, a model was developed for mines where the overlying seam has been previously mined. The regression model is given as follows:

$$\text{INTERP} = 7.68 - 0.02\text{IBT} - 0.06\text{LSEP} \\ + 0.27\text{LST.}$$

This is also a statistically significant model (F -value = 8.01) and has less

variance inflation ($p = 1.12$). These models demonstrate that interaction-controlling factors for mines using overmining methods are different from those using undermining methods. In this

MODEL ANALYSIS

To extend the information gained from case studies, analysis of various interaction phenomena has been carried out using body-loaded photoelastic and finite element models. These two modeling methods are concentrated in stress analysis in the innerburden for undermining conditions.

PHOTOELASTIC STRESS ANALYSIS

Photoelastic models were used to evaluate stress phenomena in close proximity to the excavation since they could best incorporate the effects of layering and bed separation, which is extremely difficult to accomplish using numerical methods. Peng (2) predicted that the load approaches the normal background value at a distance of approximately four times the pillar width below the floor. Ehgartner (7) found that the distance stress was transferred depended on the nature of the rock below the pillar. In particular, he found that low-modulus stratified materials tended to increase the distance through which stress is transferred, while stiff isotropic materials had the opposite effect. Results of this model showed that the zero-influence bulb could extend as deep as eight times the pillar width in a highly stratified material (fig. 6).

FINITE ELEMENT ANALYSIS

Finite element analysis was used to evaluate the influence of thickness and location of sandstone layers in the innerburden and overburden on the size and shape of the pressure arch around the lower opening. This information was, in turn, used to locate the area in which the upper seam arch intersected the lower workings. Hudock (16), modeling in isotropic overburden, found that the modulus of the material had little influence on

latter model, the important variables are innerburden thickness, lower seam extraction percentage, and lower seam thickness (15).

arch dimensions or abutment pressures. A high-modulus layer above an excavation, however, can affect subsidence throughout the formation (2, 17). The effects of opening geometry and material properties on the stability of interactive excavations in stratified rock were also evaluated using finite element methods. The influence of pillar location, Young's modulus, and Poisson's ratio were determined. This system was modeled with four-node isoparametric, quadrilateral, linear elastic, plane strain elements. The standard finite element routine was modified to include the calculation of the factor of safety at the center of each element based on the Mohr-Coulomb failure criterion as outlined by Wang (18) and Haycocks (3). The results from this work showed that increasing Poisson's ratio resulted in improved stability in the roof of both upper and lower openings for all innerburden spacings. The shear stress at the pillar edge in the roof of the lower opening and the floor of the upper opening is also

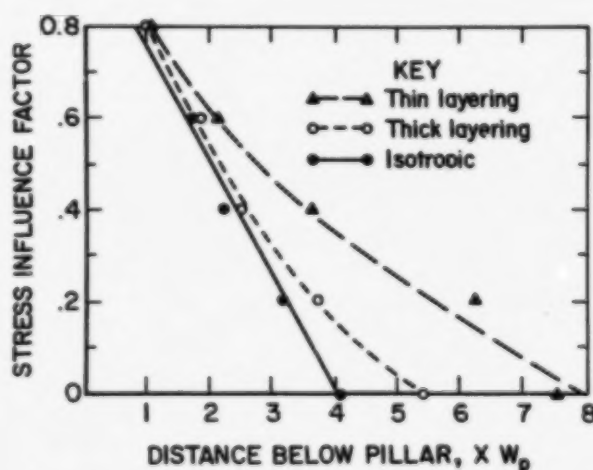


FIGURE 6.—Effect of the degree of layering on the size and shape of the pressure bulb based on photoelastic models. Adapted from Ehgartner (7).

decreased, indicating less chance of massive inner seam shear. Alternatively, increasing Young's modulus raised the tensile stress in the roof above both openings but decreased the maximum compressive stress in the pillars of both seams. The shear stress in the roof of the lower opening and the floor of the upper opening is slightly decreased in this condition.

Although the finite element modeling method has its limitation due to the

idealization of true conditions, it was felt that this method was useful in demonstrating trends as to the influence of variables on opening stability, even though the calculated stresses and factor of safety were only valid for comparison purposes. The incorporation of these model studies with field-observed data can provide realistic design criteria and methods for evaluating a wide variety of mining geometries and geologic settings.

ANALYSIS OF UPPER SEAM SUBSIDENCE

When dealing with interaction problems occurring during overmining operation, a different approach is used. Empirical methods were used to classify the upper seam structure and to predict upper seam stability.

METHODS OF PREDICTION

Based on statistical analysis, three methods have been proposed for evaluating upper seam stability, depending upon the application needs and type of input data available. The first method calculates a damage factor (8); the second, or M-index method, uses the critical innerburden thickness (19); and the third, which apparently holds the most promise, is the subsidence factor method.

Plots of field data show that the overall damage rating can be related to the maximum subsidence (8). Figure 7 shows the relationships between damage factor and maximum predicted subsidence at the level of the upper seam for the various roof conditions. Roof structural conditions can be evaluated using the various geomechanics classification systems, taking into account factors that are peculiar to multiple-seam operations. An index can be assigned to each class of roof as follows:

- 1 = Very good roof.
- 2 = Good roof.
- 3 = Fair roof.

4 = Poor roof.

5 = Very poor roof.

The damage rating can then be written as

$$DF = a_1 + b_1 S_{max} + a_{11} x_1 + a_{12} x_2 + a_{14} x_4 + a_{15} x_5,$$

where DF is damage factor and x_j , $j = 1, 2, 3, 4, 5$ are indicator variables representing the type of roof and are defined as follows:

$x_j = 1$ for type j roof,

$x_j = 0$ for all others.

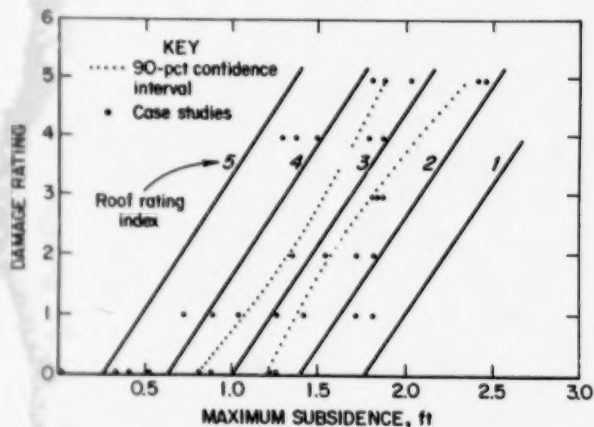


FIGURE 7.—Nomogram for predicting upper seam stability. Adapted from Webster (8).

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Precise determination of the exact mathematical relationships depends on the geological and spatial factors.

STABILITY OF UPPER SEAM STRUCTURES

Of all possible disturbances that can be caused to the upper seam, the most frequently encountered difficulty has been damage to the roof. Although pillar load increases when mining crosses the gob line to the solid side of the lower seam, as recorded by Chekan (20), the increase is usually not large enough to cause any major problems. One exception is the situation in which pillars in the upper seam are located over remnant pillars of substantial size left in the lower seam, which transfers high stresses through the innerburden to the upper seam.

COMPUTERIZED EVALUATION OF CLOSE SEAM INTERACTION

Because of the complexities and numerous steps necessary in multiple-seam stability evaluation and design, the process was incorporated into software packages available for use on a personal computer. The aim of these programs is to provide engineers with means of determining whether any difficulties will be encountered and of what type they may be when mining in a specific condition.

PROGRAM DEVELOPMENT

The computer program USEAM is used for undermining conditions and is organized into two modules: the input module and the calculation module. The input routine consists of a series of menus and an editor for entering or changing data values. This module also provides for storage of the data on diskette for later modification and use.

The calculation module uses the data supplied by the input routine to predict when strata control problems may be expected and to suggest design parameters. The calculation module is further divided into a series of subroutines, each performing a specific calculation or set of calculations. The operation of this module is outlined in figure 8.

Roof conditions in the upper seam, on the other hand, are greatly affected by the subsidence of the strata. This is because the roof is most prone to tension created by the subsidence of the strata. Therefore, the location and orientation of the roof structures in relation to the subsidence trough is a very important factor in the design of the upper seam. The two most consistent places for serious floor disturbances in an upper seam are over isolated pillars or small groups of pillars left in the underlying seam, and in areas above the line between completely mined areas, or grooves, and solid coal in the underlying seam (1). The mechanism is the transfer of stresses, as in the case of remnant pillars.

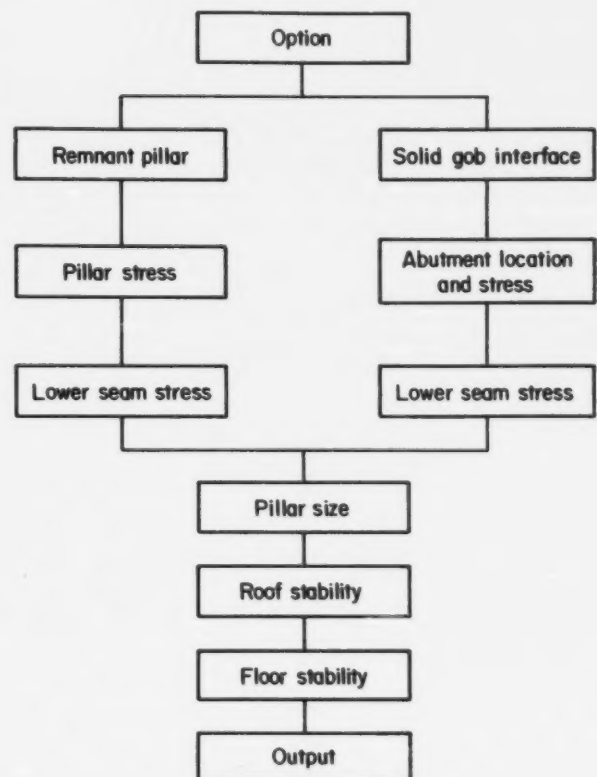


FIGURE 8.—Macro flowchart for the computational module of USEAM. Adapted from Grenoble (21-22).

Stresses for locations under a remnant pillar and abutment stress field are first calculated. The results are then used in evaluating the stability of the lower seam roof and floor, and for pillar designs. Output of the results is presented in a series of tables and design curves. Typical input to and output from the program are summarized in figures 9 and 10.

VALIDATION OF THE PROGRAM

The program USEAM has been validated by some field study data. For example, in the case reported by Stemple (1), mining operation was conducted under an arch

abutment in the upper seam. The overburden in the problem area was approximately 1,500 ft, while the innerburden consisted of a massive 35-ft-thick sandstone layer. The excess weight created by the pressure arch abutment caused the pillars in the lower seam to bump and eventually crush.

The program predicted that the average stress on the lower seam will approach 2,500 psi. From this information pillar dimensions on the order of 130 ft square are suggested for a safety factor of 1.5. Since the actual pillars were much smaller than this (25 by 80 ft), it is obvious that they could not be expected to support the load.

CONCLUSIONS

A comprehensive study has been conducted on the problems of interactions in multiple-seam operations for overmining, undermining, and simultaneous mining, integrating results from field studies, statistical analysis, and numerical and physical modeling. Important findings of this research effort are summarized below.

The majority of the interaction problems occurred in one of three modes: mining under remnant pillars, mining under gob line interface, and mining over gob line interface.

The most important controlling factors are extraction percentages, innerburden thickness, lithological structure, and time delays between operations in the two seams.

Roof structures can be characterized by taking into account the many geological factors and mining factors associated with the two adjacent seams and their innerburden, maximum subsidence, and relative location and orientation of seam workings.

Four interaction mechanisms were identified: pillar load transfer, innerburden shearing, arching effects, and upper seam subsidence. The controlling mechanism may not be the same for different mining sequences, and it may be appropriate to consider a different set of controlling factors when a particular mining sequence is selected.

Simultaneous mining is likely to cause more interaction problems than other methods and therefore should be avoided if possible.

For undermining, design guidelines have been proposed for the various mining sequences and design criteria have been assembled on a microcomputer in a user friendly, interactive package. The program has been so written that installation is easy. Driven by an interactive menu, data input and sensitivity analysis can be done at the fingertips. Most of the relevant input data can be used but are not strictly required to carry out a comprehensive analysis. In addition, a regression model has also been developed that predicts a damage factor.

To evaluate upper seam stability in overmining, three methods have been proposed for the Appalachian coalfields. These are the damage factor method, the M-index method, and the subsidence factor method. Any one of these methods can be used to predict potential hazards in the upper seam, depending upon the applications and type of available data. Nomograms have been prepared for quick accessibility for the field engineers.

Ground control in multiple-seam mining is a complex problem; however, under most conditions, interaction effects and their magnitude can be predicted with accuracy, provided sufficient information is available on the many geologic and mining

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INPUT

Mining site: Lee County, VA

Mine under: 1. Remnant pillar
2. Solid-gob interface
3. Pillars in place

Enter number: 1

UPPER SEAM DATA

Seam name: Pocahontas No. 5
 Depth of overburden (ft): 510
 Density of overburden (lb/cu ft): 140
 Mining height (in): 57
 Residual pillar dimensions (length and width, ft): 150,245
 Coal K factor: 0.27
 Roof character (10=excellent, 1=poor): 8
 Floor character (10=excellent, 1=poor): 6
 Jointing present? (y/n) y

Joint set No.	Spacing (ft)	Orientation (deg)	Dip (deg)
1	26	170	85
2	40	80	70

INNERBURDEN DATA

Average thickness (ft): 55
 Total number of beds: 8
 Sandstone present? (y/n) y

Bed No.	Thickness	Distance above lower seam (ft)
1	10	30
2	4	39

Jointing present? (y/n) y

Joint set No.	Spacing (ft)	Orientation (deg)	Dip (deg)
1	40	80	70

LOWER SEAM DATA

Seam name: Pocahontas No. 3
 Seam height (in): 48
 Roof rock characterization (10=excellent, 1=poor): 4
 Floor rock characterization (10=excellent, 1=poor): 9
 Coal K factor: 0.23
 Jointing present? (y/n) n

FIGURE 9.—Typical input data for USEAM.

OUTPUT

Mining site: Lee County, VA
 Seam analyzed: Pocahontas No. 3

Optimum pillar dimensions: 50 ft long
 55 ft wide
 70-ft centers

Optimum room width: Entries 20 ft
 Crosscuts 15 ft

Projected extraction: 51 pct

Maximum pillar load: 4,873 psi
 Average pillar load: 1,763 psi

Roof index (10=excellent, 1=caved): Worst 4
 Average 7

Floor index (10=excellent, 1=serious heave): Worst 5
 Average 8

Most likely failure mechanism: Roof shear
 Potential for cracking to upper seam (0=none, 5=certain): 3
 Probability for success: Good

Graphics options: Enter number

1. Plot of upper seam geometry
2. Plot of lower seam geometry
3. Plot of superimposed geometries
4. Contour of pillar load
5. Contour of roof index
6. Contour of floor index
7. Plot of projected failure zone in cross section

FIGURE 10.—Typical output from USEAM.

factors. Integration of the field studies, statistical analysis, numerical and physical modeling, and the help of modern computers will bring a better

understanding of the mechanisms governing interaction problems and provide guidelines for ameliorating negative effects.

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APPENDIX.--VARIABLES SELECTED FOR DATA COLLECTION

Quantitative Variables

1. COVER - Cover depth, ft.
2. UST - Upper seam thickness, ft.
3. LST - Lower seam thickness, ft.
4. USEP - Upper seam extraction percentage.
5. LSEP - Lower seam extraction percentage.
6. IBT - Innerburden thickness, ft.
7. IBSP - Innerburden sandstone percentage.
8. IBNL - Number of layers in the innerburden.
9. TIME - Delay between operations in two seams, yr.
10. INTERP - Degree of damage due to interaction:
INTERP = 0 - no damage.
INTERP = 5 - serious damage.

Qualitative Variables

11. LOCATION - Location of mine.
12. BEDNAME - Names of two seams.
13. CRT - Cover rock type.
14. USIR - Immediate roof for upper seam.
15. LSIR - Immediate roof for lower seam.
16. USIF - Immediate floor for upper seam.
17. LSIF - Immediate floor for lower seam.
18. USLP - Locations and damages in upper seam.
19. LSLP - Locations and damages in lower seam.
20. IBRT - Innerburden rock type.
21. WP - Existence of water problems.
22. MM - Mining sequence.
23. SSUBP - Surface subsidence problem.
24. LITERP - Location of interaction problems.

THE BUREAU OF MINES SUBSIDENCE RESEARCH PROGRAM

By Michael A. Trevits,¹ Roger L. King,² and Bradley V. Johnson³

ABSTRACT

The Bureau of Mines, through its Subsidence Research Program, is focusing on providing the mine operator with the ability to predict surface movements and effects on ground water as a function of mining method and geologic context. The program is designed for coal basins where high mining activity may impact land use requirements. In the long term, all coal basins and mining methods will be addressed. Data sets from several subsidence monitoring sites have been or are being collected. Data sets are now

available from the Eastern, Interior, and Rocky Mountain Coal Provinces for full-extraction mining methods (longwall and/or room-and-pillar retreat mining). At select sites, shallow-aquifer monitoring wells have also been installed to observe the effects of subsidence on the ground water system. To date, an empirical model for subsidence prediction has been generated for the Northern Appalachian Coal Region. This paper discusses the status of the Bureau of Mines Subsidence Research Program.

INTRODUCTION

The involvement of the Bureau of Mines, U.S. Department of the Interior, in mining-related subsidence probably dates back to the Bureau's inception in 1910, when a mining engineer was assigned to a study of mine filling in the northern Anthracite Field of Pennsylvania (1).⁴ The purpose of this work was to reduce the amount of mine waste resulting from imperfect mining methods, create a safe work environment, and reduce the settlement of the overlying ground surface. Results of the investigation were subsequently reported by Griffith and Conner (2).

Perhaps the longest sustained subsidence investigation ever made in the United States involved a cooperative effort between the Bureau, the Illinois State Geological Survey, and the University of Illinois during 1916-24. Reports covering the results of these studies

became classics of mining subsidence literature.

Between 1926 and 1932, subsidence studies were pursued under various sections of the Bureau's Mining Division: Ground Movement Investigations Section, Subsidence Section, and Subsidence and Ground Movement Section. During this period, subsidence was studied in connection with iron mining in Alabama and Oklahoma, and iron and copper mining in Michigan.

Although subsidence research did not constitute a distinct organizational element in subsequent years, concepts and data fundamental to an understanding of the mechanics of subsidence were developed in the course of many other Bureau investigations. Included among these were mine roof control, pillar extraction, longwall and caving methods of mining, physical properties of mine rocks, in situ stress measurements, and physical and mathematical modeling of mine structures.

The Bureau's subsidence research was focused in the Minerals Environmental Technology Program in 1978. In 1982 certain activities of this program were melded with work in mining technology to form the current Mining Technology Program under which subsidence research is presently conducted.

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⁴Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

THE PROCESS OF SUBSIDENCE

During the subsidence process, the ground surface is changed as a result of the excavation of underlying coal. The final surface profile is determined by a multitude of components: thickness of the coal extracted, width of the opening, depth of coal, lithology of the strata between the coal and the surface, mine design, mining method, and other factors. An analysis of the typical subsidence profile reveals that almost every point of the surface will be displaced both vertically and horizontally, causing zones of tensile and compressive strain along the surface. These can result in damage to surface structures.

One must also consider the effect on ground water movement in the vicinity of the affected area. Historically, mining operations have been accused of causing water wells to become dry and/or polluted. However, such local effects may be minor compared with the potential problems that can occur in the Western and Midwestern United States. Western underground aquifers are a major source of water for populated areas; in the Midwest, surface drainage patterns are important to agriculture. It is, therefore, easily understood how changes in the ground water system could have a major impact.

SUBSIDENCE MODELS

Three types of models have gained acceptance by subsidence researchers--physical, analytical, and empirical. Physical models have been used in an effort to develop basic understandings of the subsidence process. Examples of physical models include centrifuges, sand models, and scale models of the overburden. Various researchers have claimed different findings, but the drawbacks to such models (accuracy and expense) have precluded widespread acceptance of this approach. However, basic insight into the subsidence process may be obtained from physical models.

With the advent of computers, analytical models have become more practical to use in subsidence research. This tool, in conjunction with sophisticated software (finite-element programs), permitted researchers to analyze many more cases in a given time than they could using physical modeling. The major drawback to analytical models is that the researcher must mathematically describe a physical process for which very little is known. Another major disadvantage is that the analytical model requires detailed

geotechnical information about the overburden that may not be readily available to accurately predict the subsidence profile. Such unknowns result in predictions that do not give the required degree of accuracy.

The empirical model has gained the most worldwide acceptance as the approach for subsidence prediction. Empirical models are based upon the fact that given enough field data (final subsidence profiles), a mathematical function can be written to describe a characteristic profile. This approach, of course, assumes that within a given area the important functional parameters (angle of draw, subsidence factor, etc.) are reasonably constant. If this does not happen to be the case, adjustments to the empirical model must be made to make it more or less universal. The major drawback to empirical models is that they require a substantial amount of field data before an accurate mathematical function can be derived. However, for the mine operator, this approach should be the easiest to utilize in predicting mine subsidence.

STATUS OF BUREAU OF MINES RESEARCH

The short-term strategy is to focus the research program into the coal basins where high mining activity is coupled

with present land use requirements. These areas are delineated via consultations with industry and the Office of

Surface Mining. In the long term, all coal basins and mining methods will be addressed.

Data sets from several subsidence-monitoring sites have been or are being collected. The purpose of these data sets will be to develop and/or validate the

predictive models being proposed by Bureau researchers. Initial data sets are now available from the Eastern, Interior, and Rocky Mountain Coal Provinces for full-extraction methods (longwall and/or room-and-pillar with retreat mining).

ALTERNATIVE SURVEY SYSTEMS

Since the Bureau's Subsidence Program is concentrating on the development of empirical model(s), the collection of data sets from a variety of locations is required. A study at a mine site normally requires the installation of hundreds of survey monuments, followed by numerous conventional ground surveys over a period of time. If these tasks are being carried out at a number of mine locations, the data collection scheme can become prohibitively expensive. To compare alternate surveying capabilities, a test grid was constructed at the Bureau's Pittsburgh Research Center, Pittsburgh, PA. The grid of monuments was established in an area that was not being undermined to eliminate the variable of surface movement (3).

Conventional and high-technology surveying systems including an electronic distance meter-theodolite-level (EDM-theodolite), an automatic recording infrared laser tacheometer, a global positioning system satellite surveyor (GPS), aerial photogrammetry, and prototype inertial surveyor were deployed over the grid during a 1-month period.

Highlights of the systems used follow:

The inertial system is extremely portable and can be placed in surface vehicles and aircraft. Sources of survey error include accelerometer measurement caused by thermal effects, platform drift rate caused by vibration variations and thermal transients, and environmental effects such as variations in the earth's gravitational field and temperature variations. Survey data for the Bureau study

could not be used owing to a computer malfunction.

The GPS system, when completed, will be able to determine receiver position instantaneously. Constraints are the high amperage required to power the field system, sensitivity to temperature variations, and site selection. This system was used to accurately determine the position of the control monuments, but the expense and time to survey the grid of monuments at the Bureau's study site precluded its use over the entire grid.

The EDM-theodolite system is portable, rugged, and relatively simple to operate. The system is sensitive to windy and/or rainy conditions and is subject to errors due to hand tabulations and computations. This system was used as the base for the Bureau's study because it is the most commonly used method for making subsidence measurements in the field.

The tacheometer system is completely automated for rapid recording, computing, and data generation. The system is subject to the same environmental constraints as the theodolite system.

The aerial photogrammetry system can be used for inaccessible areas. All data are gathered simultaneously, providing a permanent record that can be reevaluated. Surveys cannot be conducted during inclement weather, and data for control points must be visible on the photographs.

The results of the Bureau's study showed that the three-dimensional displacements from the base (EDM-theodolite) were almost identical for the tacheometer (0.25 ft) and photogrammetry (0.24 ft).

GROUND SURFACE AND STRUCTURE RESPONSE

In 1985, the Bureau of Mines initiated a cooperative mine subsidence research program with the State of Illinois (4-6). The purpose of this effort is to jointly

fund research to develop guidelines for underground mining methods that could maximize coal recovery while preserving farmland productivity.

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In 1985, the research effort evaluated and selected sites at operating mines that best fit the program's goals. Initial surface and subsurface studies include characterizing the response of overburden to subsidence, assessing the impact of subsidence on ground water, monitoring and characterizing subsidence profiles, collecting available data to build data bases on subsidence and rock mechanics, testing floor materials in mines, and evaluating crop production in subsided areas.

Because much of the farmland in Illinois is flat with water table depths of 2 to 5 ft, differential surface ground movements could alter subsurface hydrological patterns and affect surface drainage. The Bureau of Mines is currently studying the effects of subsidence

on surface drainage patterns to determine critical slope changes that may cause drainage problems detrimental to crop production. Such differential movements also can cause damage to structures and utilities, not only in Illinois but potentially in any area affected by underground mining.

By the end of 1984, more than 1,500 claims for structural damage (of which 25 pct were attributed to mine subsidence) had been filed with the Illinois Mine Subsidence Insurance Fund. To better understand how surface ground movements from high-extraction coal mining affect a residential foundation, the Bureau of Mines built two foundations which are currently being monitored for vertical, horizontal, and differential movements.

EFFECTS ON HYDROLOGY

To properly characterize the effects of mining-induced subsidence on the local hydrological system, a coordinated data collection scheme must be constructed. All monitoring points (weirs, wells, etc.) should be installed in advance of mine development in the area of study. It should be noted that these data will represent base conditions, and enough preliminary data should be collected to allow for seasonal fluctuations and well stabilization if applicable.

One such study was recently completed by the Bureau of Mines at a mine site in western Pennsylvania (7). The purpose of this work was to observe the effects of longwall mining in five water wells which typified local domestic water systems (fig. 1). Two very small streams and a spring located in the vicinity were also monitored to establish flow rate characteristics. The overburden above the coal unit being mined (Pittsburgh Coalbed) ranged from 750 to 1,000 ft thick.

Results of the study showed that there was no pronounced change in the overall quality between premining and 1 yr after

the longwall face passed the water well profile (fig. 2). Water levels in the well located near the centerline of the panel began to decline when the longwall face had approached within 500 ft of the well. The water level continued to fall, and the well went dry about 2 months after the face had passed beneath it, at which time the face was approximately 500 ft beyond the well location. The water levels in the two wells located 100 and 300 ft outside the rib line of the panel declined some 15 to 30 ft as a result of mining but recovered to near premining levels about 10 months after the longwall face passed by. The well located more than 500 ft outside the rib line of the panel showed no detectable change in fluid level as a result of mining. No evidence of mining effects on the small streams or springs located within 1,200 ft of the panel could be detected.

It should be noted that the results of this study apply to the local conditions of this mine site, which may differ from conditions at other sites.

SUBSIDENCE MODEL

The specific lithology over coalbeds in the Northern Appalachian Coal Region, created by highly resistive limestone and sandstone units with relatively

shallow overburden, has precluded the use of predictive methods developed for European mining conditions. Bureau of Mines work to date has resulted in the

DISTANCE OF FACE FROM
WATER WELL SECTION, ft

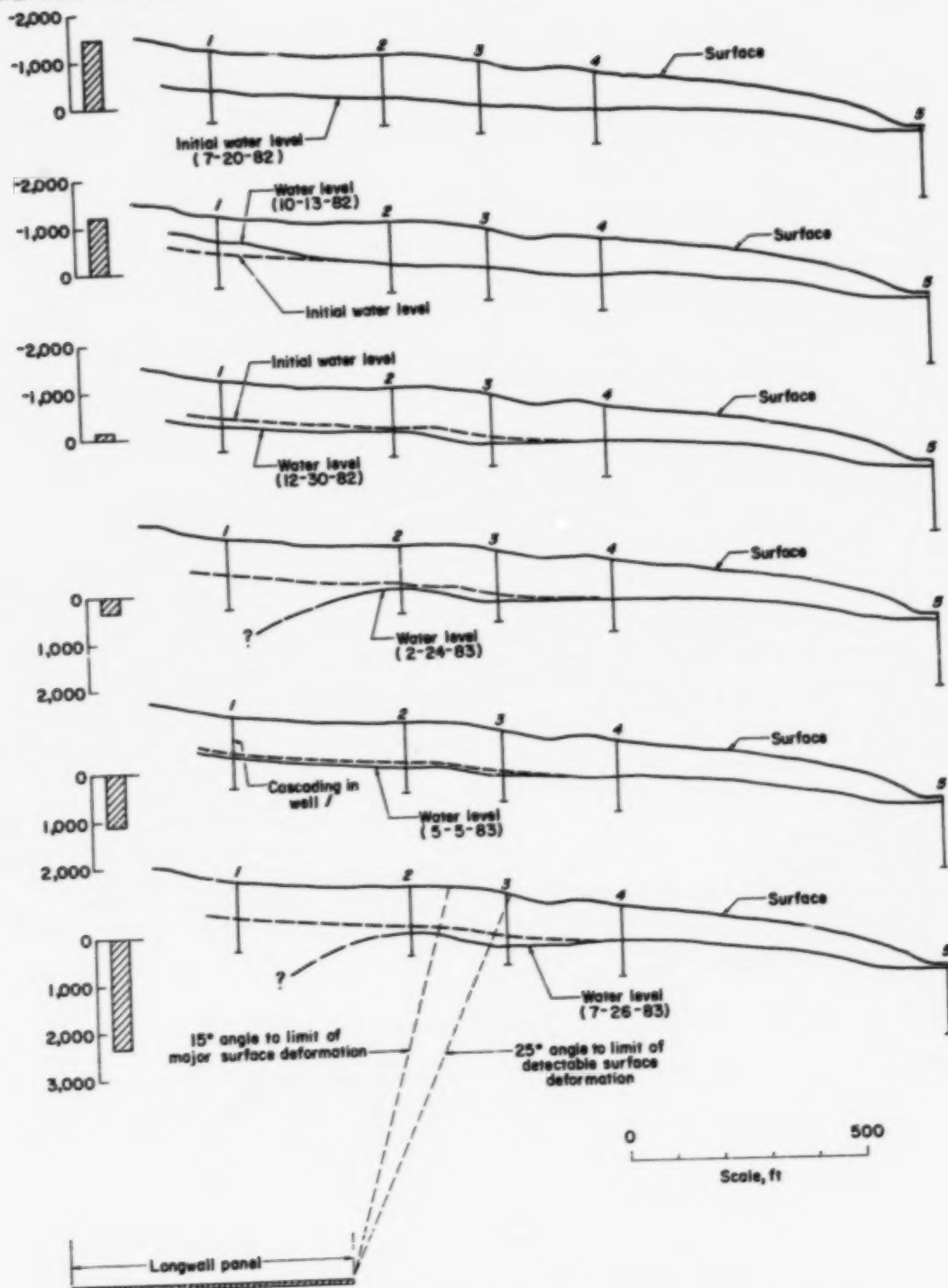


FIGURE 2.—Relation of water levels to longwall face advance.

Although very good results for the model have been obtained through sensitivity tests, the model must be considered preliminary. The model has been constructed using a limited amount of field data for relatively similar mining conditions. Its applicability in other mining areas must be tested.

To facilitate the use of the predictive model, a program was written in BASIC language for use on personal computers. The program requires approximately 48K of memory and has been constructed so that hard copies of the subsidence predictions are possible. The inherent beauty of the predictive model is that it can be operated by individuals with little or no previous knowledge of the theory of subsidence, it is simple and fast in comparison with existing predictive methods, and it eliminates the use of inaccurately estimated functional parameters, e.g., maximum subsidence, location of the inflection point, etc. necessary with existing predictive methods (fig. 3).

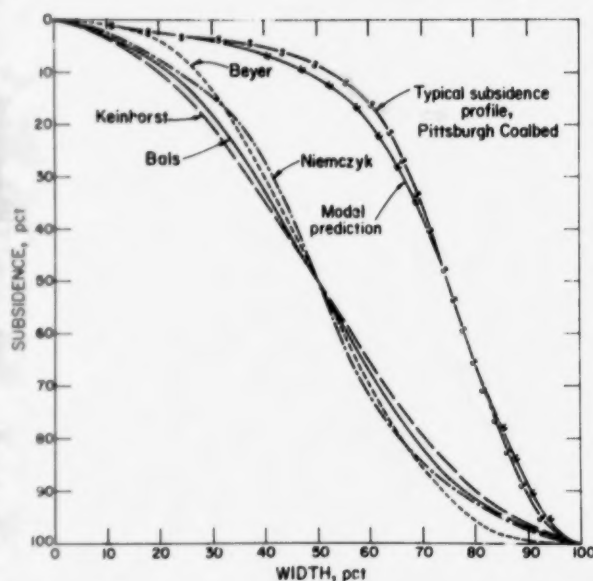


FIGURE 3.—Comparison of field data, subsidence model prediction and data obtained using various European methods.

SUMMARY

The Bureau's Subsidence Research Program has been constructed to address nearly every aspect of the phenomenon of subsidence and the results of ground movement. The program is currently focused on high-coal extraction systems; however, in the long term all coal basins and mining methods will be investigated. Work completed to date includes preliminary observations of the effects of long-wall mining on shallow water sources, delineation of the impact of mining on

primary and secondary land use, and development of a prediction model that is fast and simple to use. Future research is scheduled to quantify the effects of mining on the hydrological system in a variety of geologic environments, surface areas (including structures) will be instrumented to observe movement as mining progresses beneath, and the prediction model will be modified and/or updated for coal basins outside the Northern Appalachian Coal Region.

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SUBSIDENCE OVER CHAIN PILLARS

By P. W. Jeran¹ and V. Adamek²

ABSTRACT

Subsidence over two or more adjacent longwall panels and the intervening chain pillars was monitored by the Bureau of Mines at four mines in the Northern Appalachian Coal Basin. The magnitude of the subsidence over the chain pillars ranged from 0.06 to 1 ft. The width of the chain pillars affects the shape of the subsidence curve. Wider chain pillars yield a wider area of minimum subsidence. Comparison of the field-measured

subsidence with precalculated subsidence over the chain pillars indicates a range of pillar deformation. The data show that at three of the sites additional subsidence was induced over the first panel by the mining of the second panel. Curves of the additional subsidence are similarly shaped for these sites. This indicates that with sufficient data a model to predict subsidence over chain pillars could be developed.

INTRODUCTION

Early monitoring of subsidence over room-and-pillar workings showed that some exhibited fairly regular subsidence, while others were highly irregular. The irregular subsidences were thought to result from the incomplete extraction of the pillars where anything from stumps to entire pillars were left depending on local mining conditions.

When the Bureau of Mines began a project to develop a method of subsidence prediction, it was reasoned that data obtained from monitoring over longwall operations would be the least complicated and therefore the easiest to interpret the process of subsidence in this country. Longwall mining removes all the coal and any observed surface subsidence irregularities are therefore related to the response of the overburden to that removal. With room-and-pillar mining, there would always be the question of what portion of the subsidence was due to overburden movement and what was caused by incomplete mining.

The monitoring of selected longwall panels has shown that subsidence above the centerline is not uniform (1).³ The

lack of uniformity may be due to local stratigraphic inhomogeneities or variation in the thickness of coal extracted. Sufficient field data are not available, at this time, to resolve this question.

A predictive method has been developed (2) using data from half profiles extending from the centerline of a panel out over the solid coal. The method has been proven valid for the Northern Appalachian Coal Basin (3). Using the principle of superposition, the subsidence over the chain pillars separating two longwall panels can be predicted by adding the subsidence caused by the mining of each of the two panels illustrated in figure 1. This assumes that the chain pillars resist deformation and behave in the same way they would if the second panel had not been mined. If these assumptions are not met, then the actual subsidence should exceed the prediction.

Chain pillars are used to maintain the integrity of the entries surrounding longwall panels, and there is wide variation in the dimensions of chain pillars used in this country. Some are designed to yield, while others are sized to

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

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resist movement. To provide additional support at the working place, some mine operators backfill the entries between the chain pillars. Others install cribbing or posts, and in some instances they may do nothing. Each of these support materials will behave differently when it

becomes part of the gob; hence, effects on subsidence will also vary. This paper examines the surface subsidence over the chain pillars between longwall panels at four sites. The data were obtained using standard surveying methods.

DISCUSSION

The Bureau of Mines has monitored subsidence over two or more adjacent panels wherever possible. Since this is a multiyear task, great care must be taken to assure long-term stability of survey control points. Any undetected movement of the control points can yield erroneous data on ground movement. Data from four study sites were chosen based upon data reliability as well as difference in mining geometry and measured subsidence. While these data sets are insufficient to construct a model of the subsidence occurring over chain pillars, their study can provide some insight to the process.

The four study sites are located in the Northern Appalachian Coal Basin, southwestern Pennsylvania and north-central West Virginia. At each site subsidence was monitored over at least two longwall panels and the intervening chain pillars. The main criterion for site selection was that monitoring begin sufficiently distant from mining to preclude any prior subsidence movements. All surveying was conducted using EDM and precision leveling equipment. Survey control points were established and maintained throughout the monitoring. Initialization of survey points occurred prior to any possible movement. Profiles were located

far enough from the beginning and end of each panel to eliminate any reduction in subsidence caused by these areas.

The generalized stratigraphy over each study site as illustrated by columnar sections is shown in figure 2. The sections are based on coreholes drilled at or very near each study site. The strata are typical of the Northern Appalachian Coal Basin, being composed of alternating layers of varying thicknesses of resistant rocks (limestones and sandstones) separated by shale.

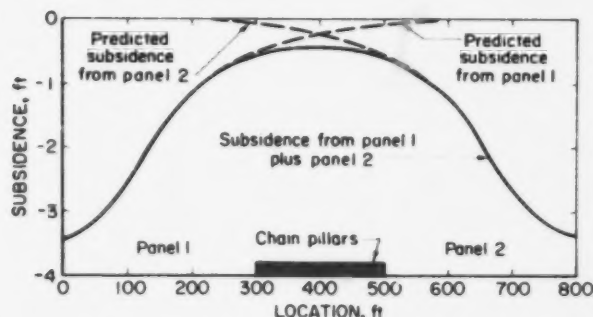


FIGURE 1.—Subsidence over chain pillars based on superposition.

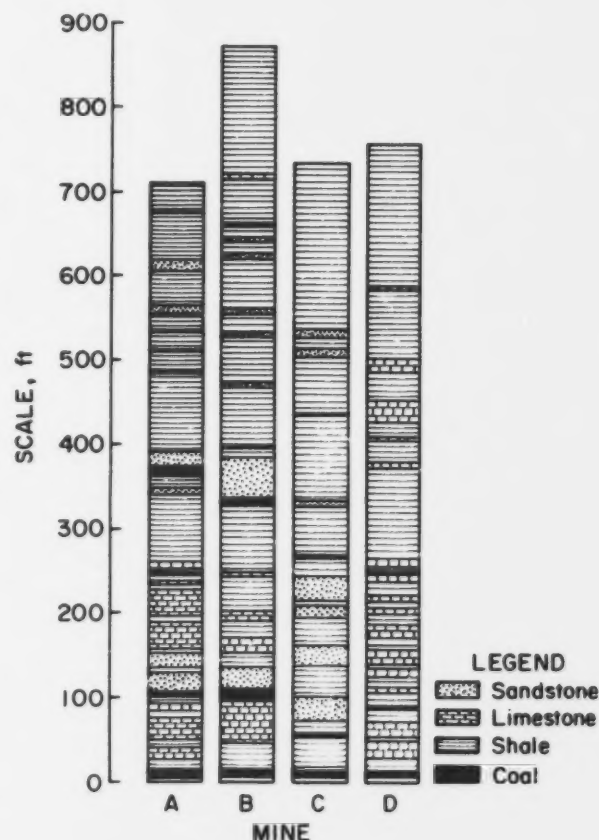


FIGURE 2.—Generalized columnar sections.

Table 1 contains a summary of the mining geometries. The panel widths range from 625 to 1,000 ft. The coal extracted averaged 5.5 to 6 ft thick. Chain pillar sizes and numbers as well as overburden thicknesses varied among the sites. Lithologic composition of the overburden was similar in that the bulk of the resistant strata at each site is within the first 300 ft above the mine.

Mine A is located in northern West Virginia on the Pennsylvania border. Three panels were monitored at this site. Each was 5,300 ft long by 625 ft wide. The panels were developed using a four-entry system driven on 100-ft centers except for the fourth entry, which was driven on 60-ft centers to accommodate a yield pillar on the gob side of the panel. The average extracted height was 6 ft. The overburden ranged from 700 to 950 ft thick over the first two panels. It exceeded 1,100 ft over the third panel. The lithologic composition of the overburden is shown in figure 2. The first 250 ft above the mined coalbed contains most of the resistant strata in the column.

Mine B is located in southwestern Pennsylvania. The panels were 630 ft wide. Panel 1 was 4,700 ft long, and panel 2 was 5,650 ft long. The panels were developed using four entries driven on 90-ft centers. The total width of the chain pillars and entries was 280 ft. The overburden ranged from 785 to 890 ft thick. The columnar section (fig. 2) shows resistant strata between 30 and 190 ft above the coalbed that was mined and a 45-ft-thick sandstone 135 ft above that. An average 6 ft of coal was extracted from these panels.

Mine C is located in north-central West Virginia. Panel 1 was 1,000 ft wide, and panel 2 was 950 ft wide. They were separated by a pair of chain pillars 62 ft wide. Total width of the chain pillars and entries was 160 ft. The overburden ranged from 665 to 690 ft thick. Most of the resistant strata are between 60 and 235 ft above the mined coalbed (fig. 2). The extraction thickness averaged 5.5 ft.

Mine D is located in the panhandle of West Virginia between Ohio and Pennsylvania. The panels were 605 ft wide by 3,000 ft long. They were developed using a four-entry system. Two of the three chain pillars were 85 ft wide, and one is 65 ft wide. The total width of the chain pillars and entries was 280 ft. The overburden ranged between 510 and 660 ft thick; its composition is shown in figure 2. The first 250 ft above the coal contain most of the resistant strata. The extraction thickness averaged 5.5 ft.

The Bureau subsidence prediction model was used to determine the amount of subsidence caused by the mining of each panel. Applying the principle of superposition, the subsidence resulting from the mining of each panel was added together to obtain the subsidence over the the chain pillars between the adjacent panels at each mine. Comparing these data to the field measurements (figs. 3-6) showed only mine D in close agreement with the model. The other mines show that subsidence greater than predicted by the model had occurred. Given that the model predictions are reasonable, it must be concluded that at three sites the overburden reacted as if the chain pillars had deformed.

TABLE 1. - Mining geometries

Mine	Overburden thickness, ft	Panel width, ft	Coal height, ft	Chain pillar	
				Centerline to centerline	Total width, ft
A.....	680-950	625	6.0	{ 2 at 100 ft 1 at 60 ft }	260
B.....	745-910	630	6.0	3 at 90 ft	280
C.....	660-710	1,000	5.5	2 at 80 ft	160
D.....	510-660	605	5.5	{ 2 at 100 ft 1 at 80 ft }	280

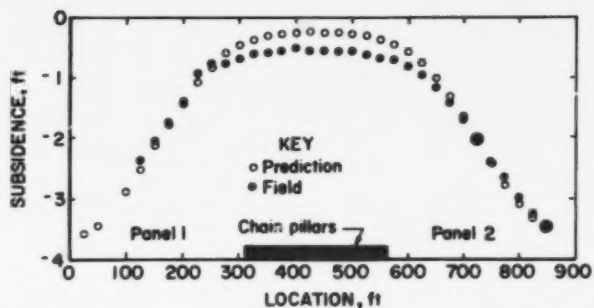


FIGURE 3.—Predicted and measured subsidence over chain pillars at mine A.

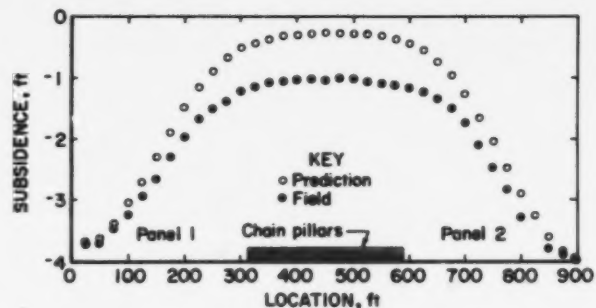


FIGURE 4.—Predicted and measured subsidence over chain pillars at mine B.

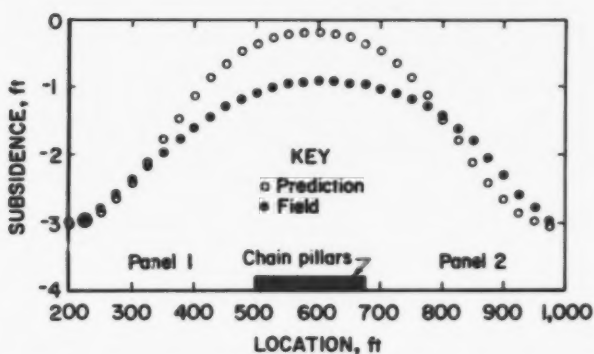


FIGURE 5.—Predicted and measured subsidence over chain pillars at mine C.

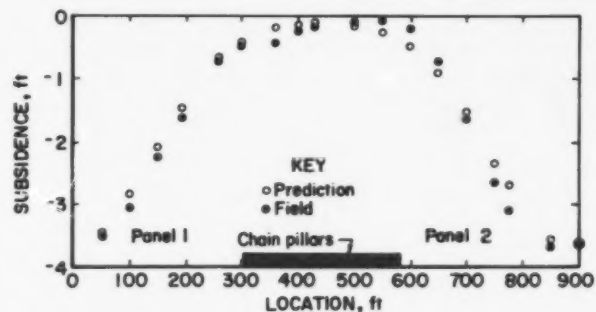


FIGURE 6.—Predicted and measured subsidence over chain pillars at mine D.

SUBSIDENCE CALCULATIONS

Subsidence can be calculated from the product of the extracted thickness, the subsidence coefficient, and the efficiency coefficient. Based on the Bureau model, the subsidence coefficient is constant beyond the edge of the panel. Therefore, the difference between the predicted subsidence and the field measurements is proportional to the product of the efficiency coefficient and a change in the height of the chain pillars. The efficiency coefficient can be calculated using an algorithm based on Bals' Theory (4). The deformation of the chain pillars expressed as a reduction in thickness can be calculated from the differences between the predicted and measured subsidence divided by the efficiency coefficient. Figure 7 illustrates the results of these calculations for each site. The results show a logical progression from no reduction in pillar thickness at mine D to the largest reduction at mine C. With increasing average reduction in thickness of the pillars, the edges appear more affected than the center. The mechanics of this apparent pillar deformation are unknown. Possible causes are the punching of the chain pillars into the floor rock, the crushing of the chain pillars, and the movement of

the roof rock into the voids surrounding the pillars.

At mines A, B, and C, where the mining of the second panel caused additional subsidence to occur over the first panel, the difference in subsidence between the two shows similar trends (figs. 8-10). At these sites the additional subsidence decreases linearly with distance from the chain pillars toward the first panel. This indicates that with sufficient data sets it should be possible to create a predictive model for subsidence over chain pillars. The data available at this time, however, do not allow the determination of the overburden thickness, panel width, and chain pillar configurations that will prevent the additional subsidence occurring over the first panel when the second panel is mined.

The time factor associated with chain pillar stability is also unknown. Should the chain pillars or the cribbing or other supports deteriorate, then additional subsidence should occur. Therefore, it appears that reducing the size of chain pillars in the gob would assure long-term surface stability. Ideally, the elimination of chain pillars would minimize surface deformation when the subsidence process is completed.

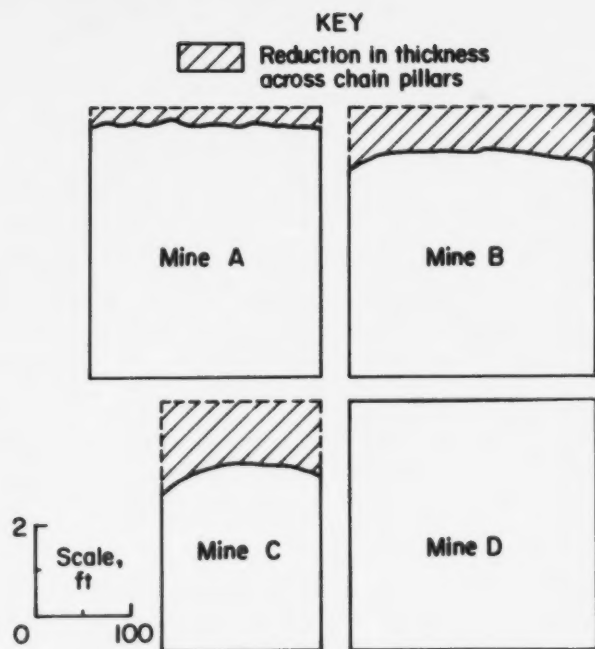


FIGURE 7.—Reduction in height across chain pillars.

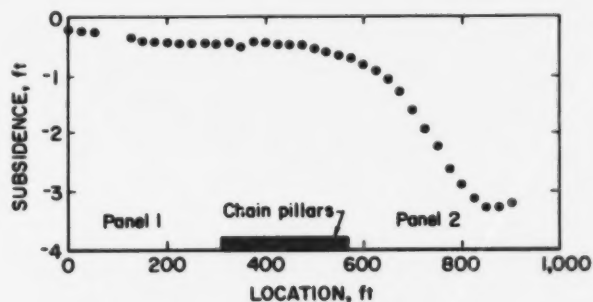


FIGURE 8.—Difference in subsidence between mining of panels 1 and 2 at mine A.

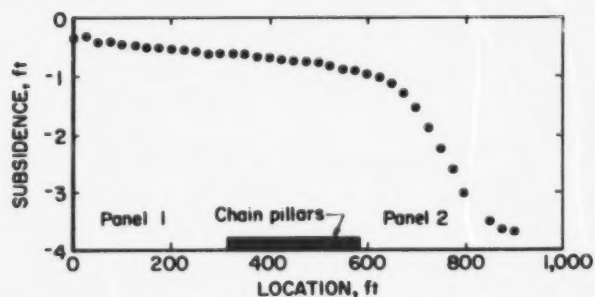


FIGURE 9.—Difference in subsidence between mining of panels 1 and 2 at mine B.

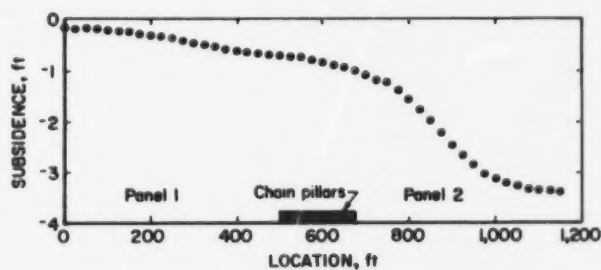


FIGURE 10.—Difference in subsidence between mining of panels 1 and 2 at mine C.

SUMMARY AND CONCLUSION

Chain pillars are the blocks of coal left in place to protect the entries between adjacent longwall panels. With the retreating of the longwall face, they become part of the gob. Their effect on subsidence is to break up the subsidence profile across several panels into separate troughs. Measurements of subsidence over chain pillars at four mines in the Northern Appalachian Coal Basin show surface movements over chain pillars ranging between 0.06 and 1 ft. The adjacent troughs subsided between 3.5 and 3.9 ft. The data indicate that minimizing the width of the chain pillars would also minimize the difference in subsidence between that occurring over the center of the panels and that over the chain

pillars. The subsidence prediction model developed by the Bureau predicts subsidence over chain pillars only when the chain pillars behave as if the second panel had not been mined. The similarity of the shapes of the curves of difference in subsidence between mining of first and second panels indicates that with sufficient data the Bureau model could be expanded to include the prediction of subsidence over chain pillars. The long-term stability of chain pillars is unknown, and the materials used to support the intervening entries are subject to eventual deterioration. Therefore the potential exists for further subsidence using the present configurations of chain pillars.

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STUDY OF DEWATERING EFFECTS AT AN UNDERGROUND LONGWALL MINE SITE IN THE PITTSBURGH SEAM OF THE NORTHERN APPALACHIAN COALFIELD

By Gregory E. Tieman¹ and Henry W. Rauch²

ABSTRACT

Dewatering effects from longwall mining were studied for a mine site in southwestern Pennsylvania as part of a research project funded by the Energy Research Center of West Virginia University with contributions by the U.S. Bureau of Mines. The mine showed evidence of dewatered streams and ground water supplies. Water sources located much above base level (major stream level) and over or adjacent to recently mined longwall panels were partly to completely dewatered, probably owing to downward leakage along

subsidence fractures. These lost waters did not migrate to the deep mine because of its thick overburden (at least 500 ft), but instead flowed laterally over confining strata to discharge at nearby streams. Many affected water supplies recovered partially, and all streams recovered fully within 1 to 3 yr following longwall mining. Spring and well recovery occurred most frequently near local stream level where newly formed springs were also common.

INTRODUCTION

BACKGROUND

The coal produced from underground mining is an integral component of the economies of the Northern Appalachian Coal Basin. Unfortunately, adverse environmental effects can be associated with underground coal mining. These effects include (1) surface damage resulting from land subsidence, (2) degradation of ground and surface water quality, and (3) stream and aquifer dewatering with associated fluctuations of the ground water levels. Federal regulations are intended to ensure the health and safety of the public and to minimize potential damage to the environment. The regulations require the mining permit applicant to "...identify the extent to which the proposed underground mining activities may proximately result in contamination, diminution, or interruption of an underground or surface source of water within

the proposed mine plan or adjacent area for domestic, agricultural, or other legitimate use" (1).³

The partial or complete loss of ground water supplies and streams can be of significant impact in rural areas of the Northern Appalachian Coal Basin. Although mine subsidence damage to a surface structure may be significant, the loss of water is often a longer term problem. For example, it is inconvenient to haul replacement water for domestic household use, and replacing lost water supplies for livestock often is uneconomical. Therefore, an assessment of the impacts of underground coal mining on nearby water supplies is important to help avoid such impacts and to ensure the optimum replacement of lost supplies.

The assessment of mining impacts upon local water supplies, particularly impacts on the longwall mining method, have been difficult to determine because of a general lack of documented evidence. The

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

longwall mining method is a nearly total coal extraction system that is accounting for an increasing percentage of underground coal production in the Northern Appalachian Coal Basin (2). In this basin there are numerous reports of negatively impacted domestic water wells, springs, ponds, and streams, but there have been few scientific attempts to quantitatively relate these reports to parameters such as type of underground mining, age of mines, topography, and mine overburden stratigraphy and thickness (1).

The overall objective of this study was to determine the impacts of two underground coal mines on streamflow and ground water supply levels. One study site (reported on in this paper as study site Y) is located in Greene County in southwestern Pennsylvania; the second site (study site Z), which is not reported on in this paper, is located in Monongalia County in northern West Virginia (fig. 1). Both of these underground coal mines utilize the longwall mining method in the Pittsburgh Coal Seam. These two coal mines were selected for study because of the cooperation of the mining companies involved as well as the similarities in geologic setting, mined coal seam, and mining methods used. Both of these mines were studied for water quantity impacts by mining in an M.S. thesis study by Tieman (3). The terrain in this portion of the unglaciated Appalachian Plateau consists of rounded hills separated by narrow stream valleys with a topographic relief of 300 to 450 ft. The mine overburden at both sites consists of shale, mudstone, claystone, fire clay, sandstone, limestone, and coal with a thickness of 650 to 1,100 ft above the Pittsburgh Coalbed. The effects of coal mining on water resources in Greene County, PA, have been generally characterized by the U.S. Geological Survey (4-5).

METHODS OF INVESTIGATION

A domestic water well and spring inventory at mine Y yielded data on 58 water



FIGURE 1.—Location map of mine study sites Y and Z.

wells and springs in February, June, and July, 1986. The inventoried water supplies were located above mine Y and within 2,000 ft of the nearest longwall or room-and-pillar mine area (fig. 2). The data collected represented the (1) static water table depth, (2) spring yield, (3) total well depth, (4) depth of well casing, (5) observed subsidence damage, and (6) hydrologic changes to the water source. The data collected were from published reports, information from mine company personnel and owners of the domestic water supplies, and from measurements made by the senior author. The depth to static water level was determined when the well top was accessible by utilizing a Soiltest electric water level meter. Spring discharge in gallons per minute was determined with a 2-gal graduated bucket and stopwatch. Stream discharge at 28 sampling points on 8 streams was measured by the authors on either or both June 6 and July 24, 1986. Streamflow was measured during baseflow conditions, with no measurable precipitation during the previous 24 h so as to have stable and easily measurable flows. Stream discharge in gallons per minute was measured using a 6-in PVC pipe and a 2-gal graduated bucket with a stopwatch. A Parshall flume with a 50-mm throat was

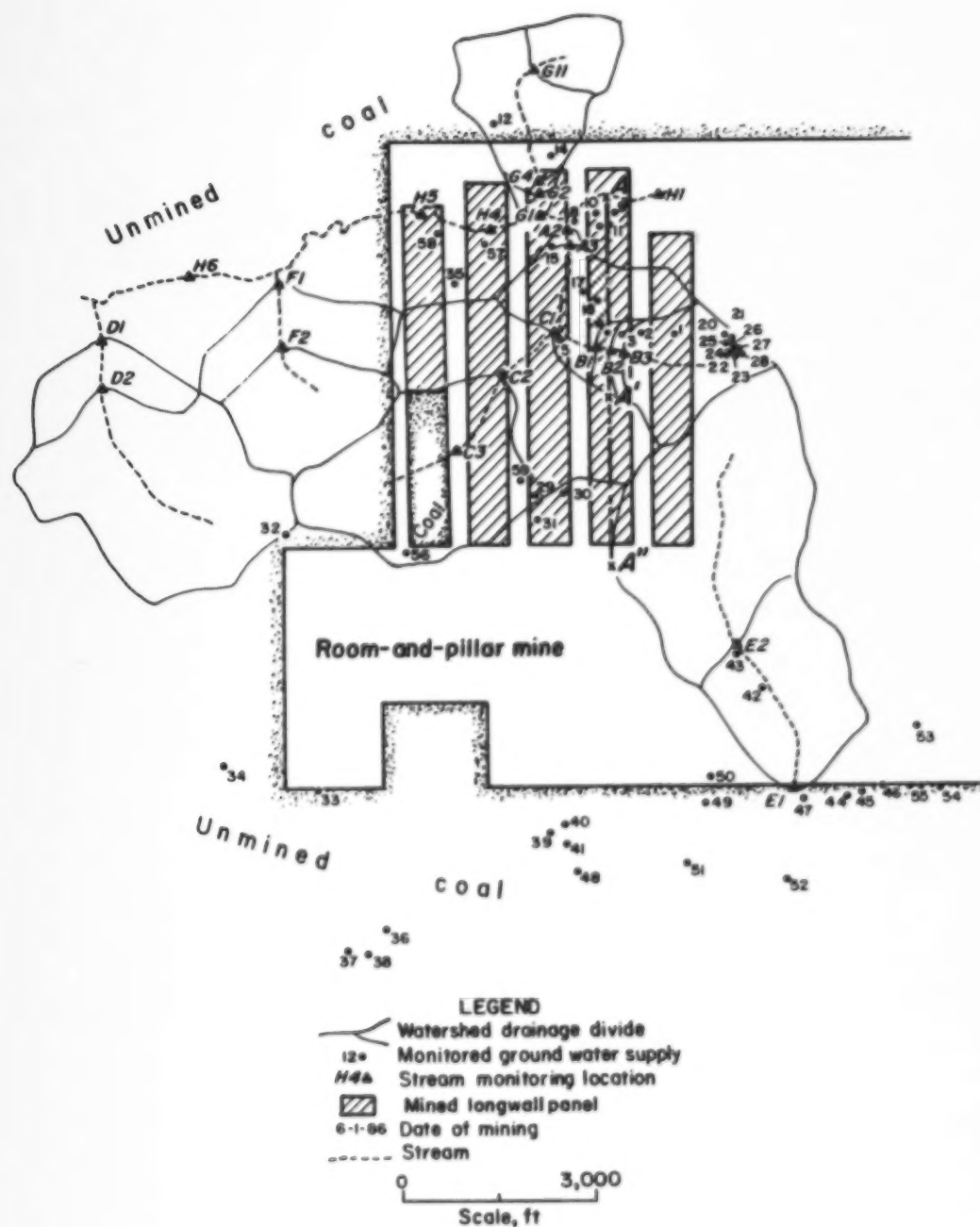


FIGURE 2.—Location of monitored ground water supplies and streams at mine site Y.

utilized for monitoring flow at stream H of mine Y owing to its relatively large size.

The locations of the sampled water supplies and stream monitoring points were plotted on the appropriate 7.5-min topographic map along with the mine outline and type of mining method used. Watershed boundaries were then determined and plotted on this map for each stream monitoring point. The watershed boundaries were constructed by first drawing two lines uphill in either direction from the stream monitoring point perpendicular to the elevation contours until the nearest watershed ridge top was intersected, and then tracing the topographic drainage divide along the ridge tops. The area of each watershed was finally determined

by utilizing a dot matrix counting technique.

The sampled wells, springs, and stream points were then plotted on various figures to determine the horizontal and vertical spacings between these features and the mine. The stratigraphic position of each water supply and stream point was also determined. Finally, horizontal and vertical spacings were measured from the sampled water supplies to both the nearest stream and the regional base level, and the topographic position of these supplies was also determined. This information was then used to determine the extent of strata dewatering and to quantify the relationships of certain hydrogeologic parameters with dewatering.

RESULTS FOR MINE Y

PREMINING HYDROLOGIC BALANCE

Hydrologic conditions existing prior to mining at the mine Y site were determined by examining nearby local streams and water supplies that were beyond the influence of mining and up to 2,000 ft away from the nearest mine. The shallow water table for hilltops was typically encountered at a depth of 10 ft, and the shallow water table under the stream valleys was encountered at depths of 10 ft and less. The depth to static level in deeper wells averaged 25 ft in the stream valleys and 100 ft on the hilltops. The most dependable water wells were developed into the Jollytown Sandstone Unit in the western portion of the mine. In the eastern portion of the mine the Waynesburg Sandstone was the most prolific aquifer. These units yielded the most dependable water supplies, especially when the water wells developed in them were located within 100 ft vertically of a stream valley. In two cases such wells

were artesian, and even nonartesian valley wells often provide enough water for four or five families each. Developed springs and water wells that are located on hill tops or upper hill slopes appear to have more variable, less dependable yields or water levels affected by seasonal changes in precipitation and evapotranspiration.

Based on the analysis of control streams in watersheds that were not undermined, the bulk of the streams can be classified as small perennial streams. During dry periods stream discharges were composed of just ground water runoff which usually sustained some measurable streamflow. Such streamflow (or baseflow) was proportional to watershed area. The normal median streamflow calculated relative to watershed area from baseflow measurements was 0.0645 gal/(min·acre) on June 6, 1986, and 0.0136 gal/(min·acre) on July 24, 1986, during a much drier period than in June.

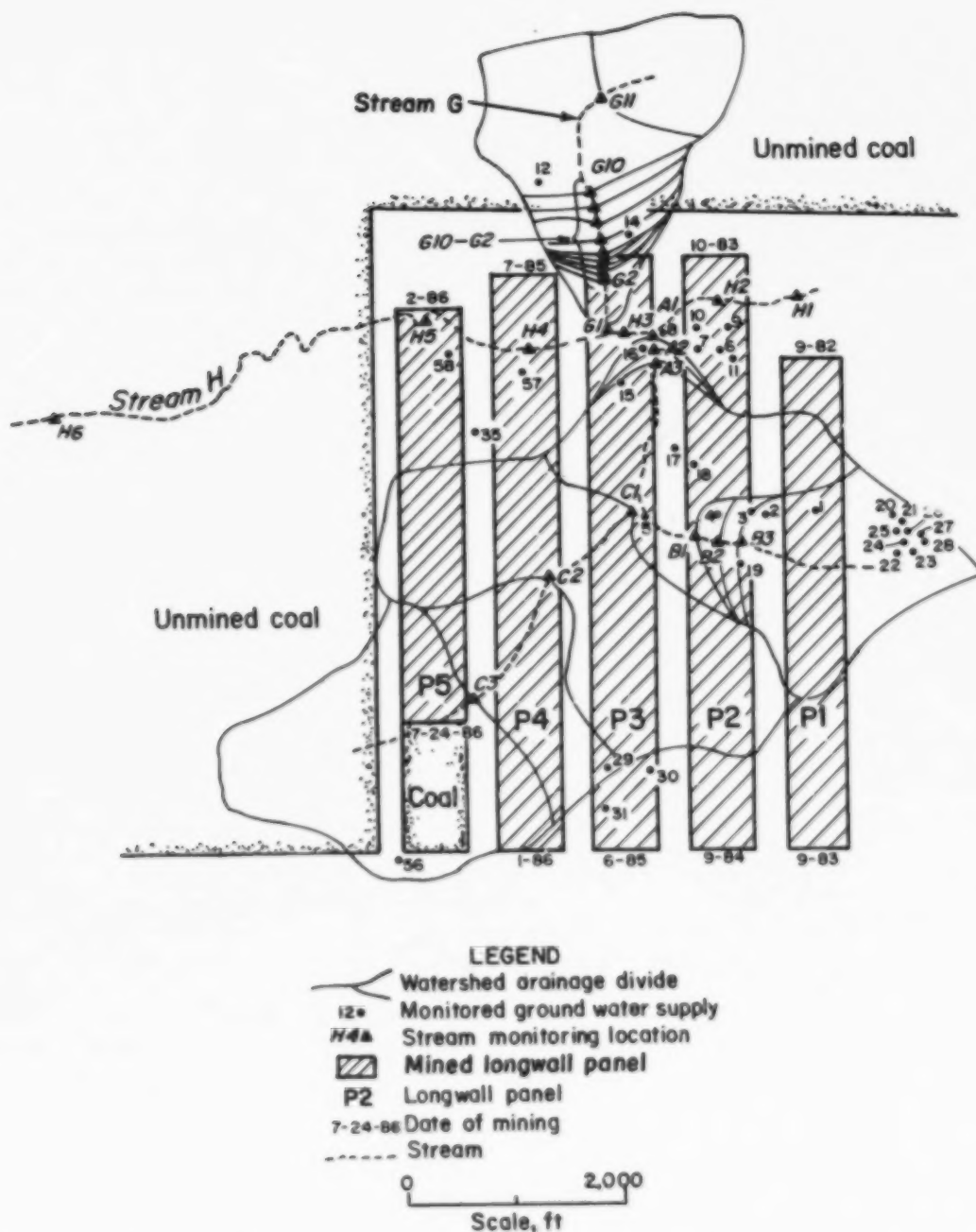


FIGURE 3.—Location of monitored ground water supplies and streams overlying or in close proximity to longwall panels at mine site Y.

MINING IMPACTS ON HYDROLOGY

Wells and Springs

Water supplies overlying or in near proximity to room-and-pillar or mine entry sections have had no reported negative impacts associated with this type of mining. This situation appears to be in response to the relatively thick mine overburden of 490 ft or more between these water supplies and mine Y.

Analysis of the water supplies developed in the subsided strata overlying longwall panels (as located in figure 3) indicated that 8 of 11 domestic water supplies that were monitored both before and after mining were partly to completely dewatered (fig. 4). The maximum amount of dewatering appears to have been more extensive near longwall panel centers, such as zone 3 of figure 4, judging by the lowest position of completely dewatered supplies. Dewatering appears to have been limited to the strata located at least 655 to 700 ft above the base of the Pittsburgh Coal. An examination of the ratio of panel width to mine overburden thickness (fig. 5) indicated that dewatering at mine Y occurred over a range of ratio values of 0.75 to 1.00. This range of values corresponded to a minimum mine overburden thickness of 655 ft (as shown in figure 4), above which dewatering was reported to have occurred.

Water supplies located adjacent to but not above longwall panels were also examined for dewatering trends. Analysis of these supplies determined that dewatering zones were present, as defined by an angle of influence (fig. 6). The angle of influence is defined as the angle between a vertical line projected upward from the edge (rib or end) of a longwall panel and a line projected to the furthest point of dewatering effects from the longwall panel (6). Eleven of 13

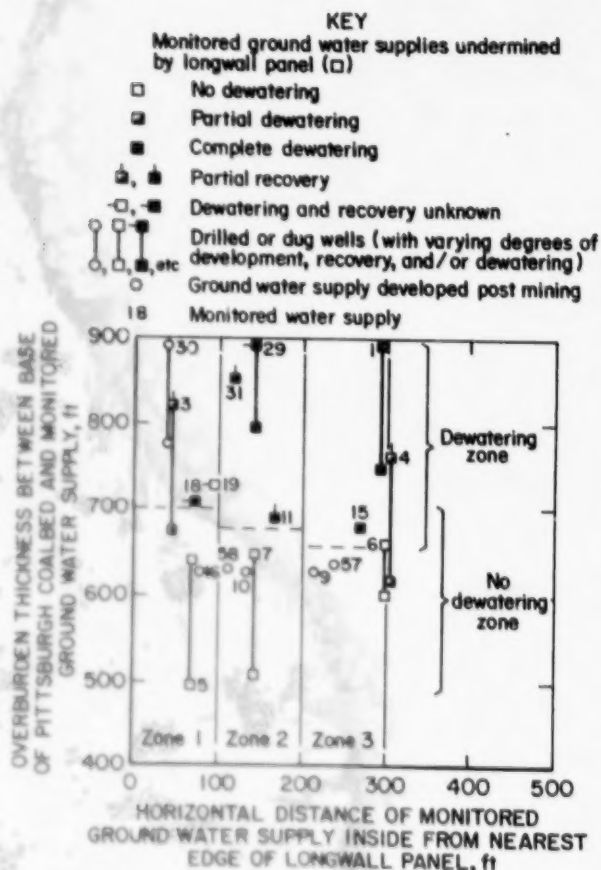


FIGURE 4.—Vertical extent of dewatering for wells and springs over longwall panels at mine site Y.

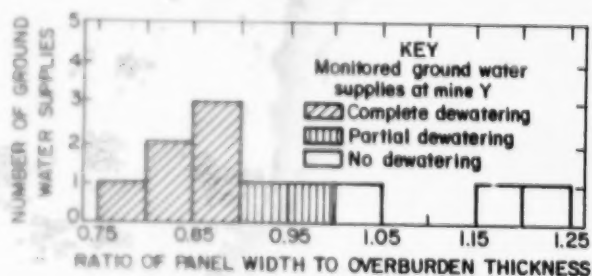


FIGURE 5.—Dewatering as a function of the ratio of longwall panel width to mine overburden thickness for monitored ground water supplies at mine site Y.

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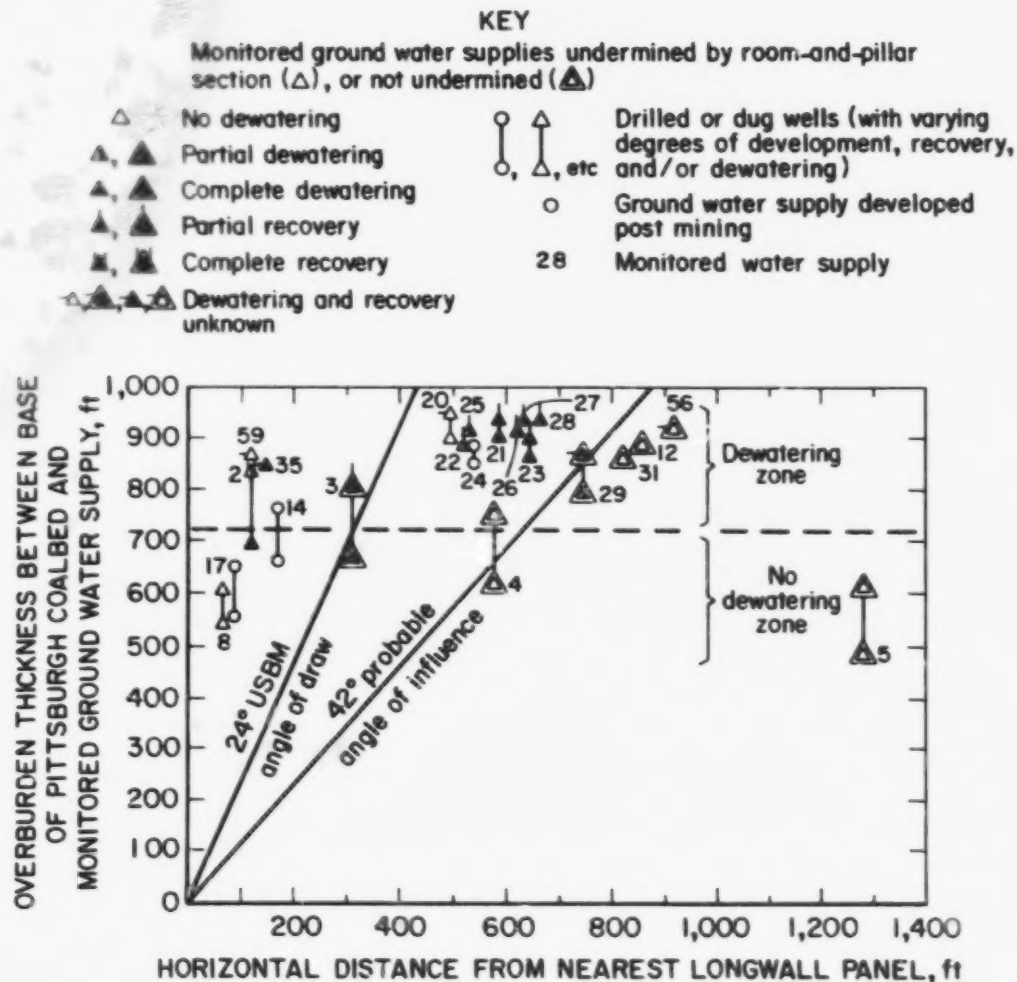


FIGURE 6.—Extent of dewatering for water supplies adjacent to longwall panels at mine site Y.

domestic water supplies that were monitored both before and after mining and located within the maximum 42° angle of influence zone were reported to have been partly to completely dewatered by the longwall mining (fig. 6). This dewatered zone adjacent to longwall panels appears to be limited to strata located at least 700 to 720 ft above the base of the Pittsburgh Seam, judging by affected water supplies that are located in or mostly in such strata.

Partial to complete dewatering of water supplies was found to usually extend no lower stratigraphically than the base of the Jollytown Sandstone (fig. 7). Stratigraphic cross section A-A'-A'' extends the length of the center of panel 2 as indicated in figure 2. The Jollytown Sandstone is situated at or just above the hypothesized regional base level that exists at the position of the major stream (stream H) for this portion of Mine Y. The extent of strata dewatering

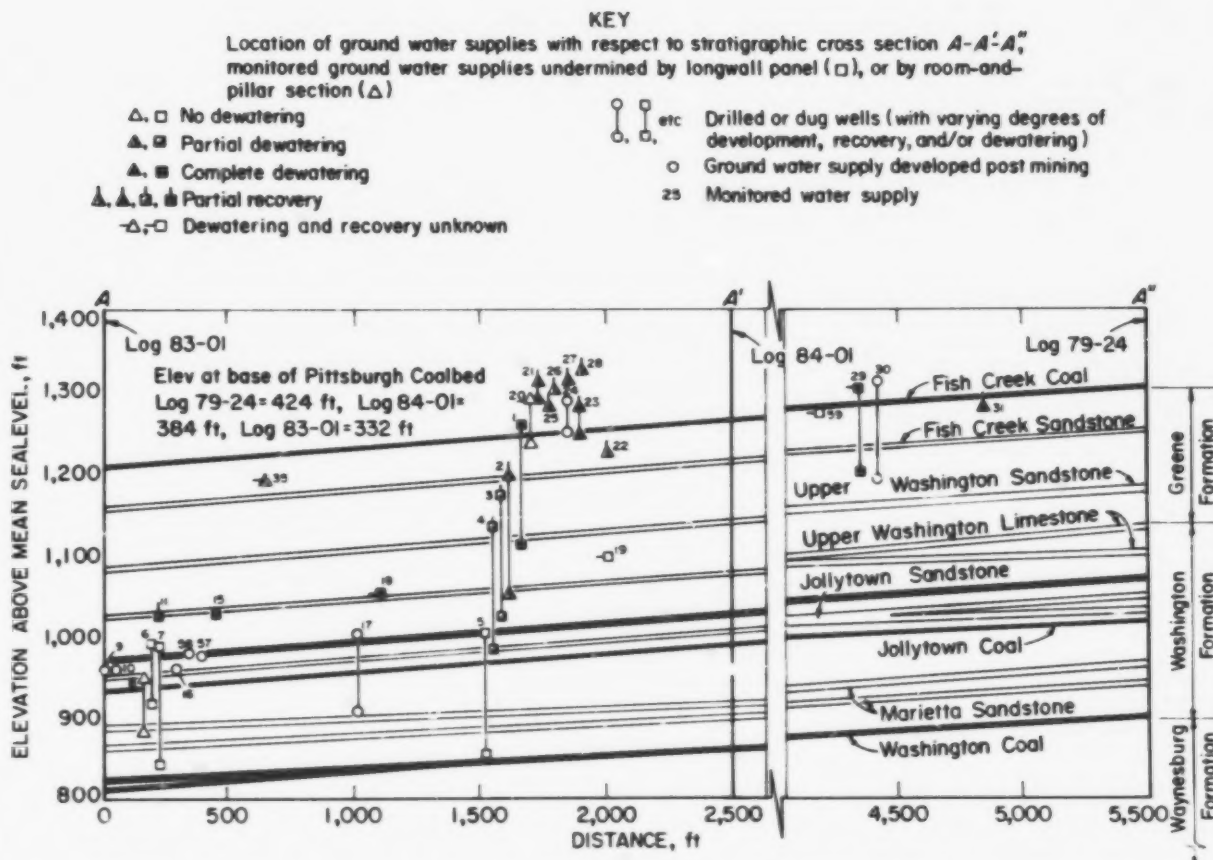


FIGURE 7 —Location of ground water supplies with respect to stratigraphic cross section A-A'-A'' at mine site Y.

with respect to regional base level is shown in figure 8 for supplies located reasonably close to stream H. Strata dewatering was found to extend down to a level located about 50 ft above the regional base level defined by stream H. Water supplies developed mostly below this 50-ft level appeared to be unaffected by the mining. Figure 9 shows that water supplies located mostly above the level of nearby perennial streams (not counting stream H) were dewatered within the 42° angle of influence zone adjacent to longwall panels. Those water supplies developed mostly below nearby

streams in elevation appear to have been unaffected by the mining. Perennial streams undermined by longwall panels, as shown in figure 10, seem to be associated with more strata dewatering than such streams not over panels (fig. 9). Dewatered supplies extend to about 20 ft or more below the elevation of nearby panel streams, with the maximum dewatering extending to 60 ft below stream elevation at the panel center position, as indicated by the water level positions measured by Moebis and Barton (1) and Walker, Green, and Trevits (7).

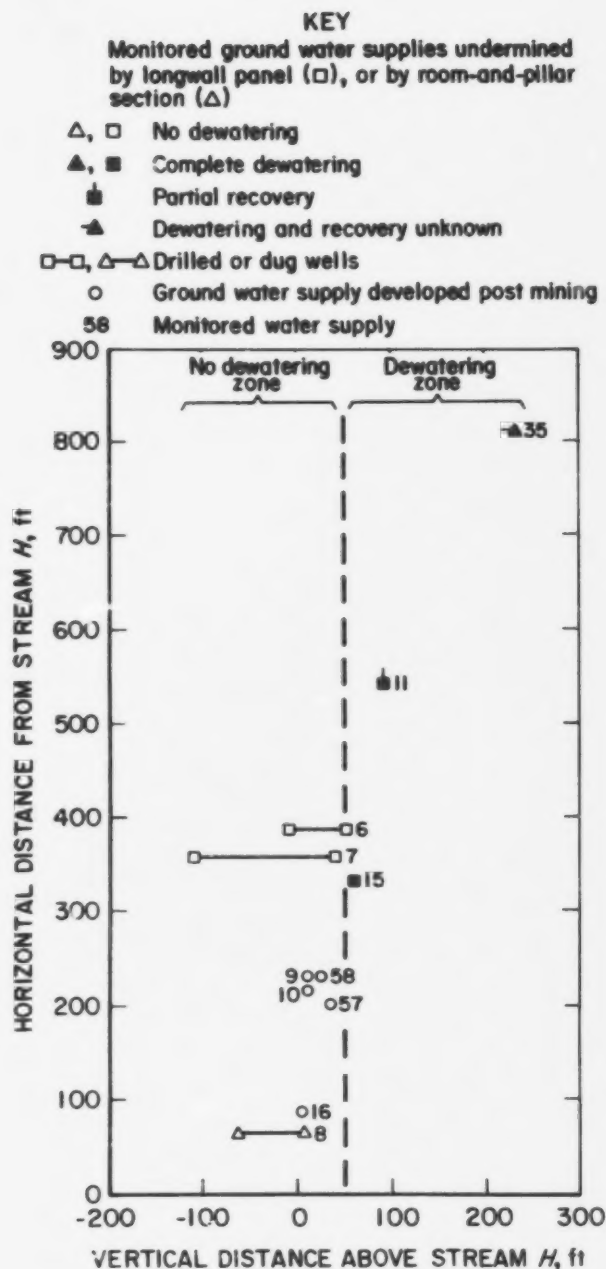


FIGURE 8.—Extent of dewatering for ground water supplies with respect to their vertical position above stream H. These supplies are nearer to stream H than to any other stream at mine site Y.

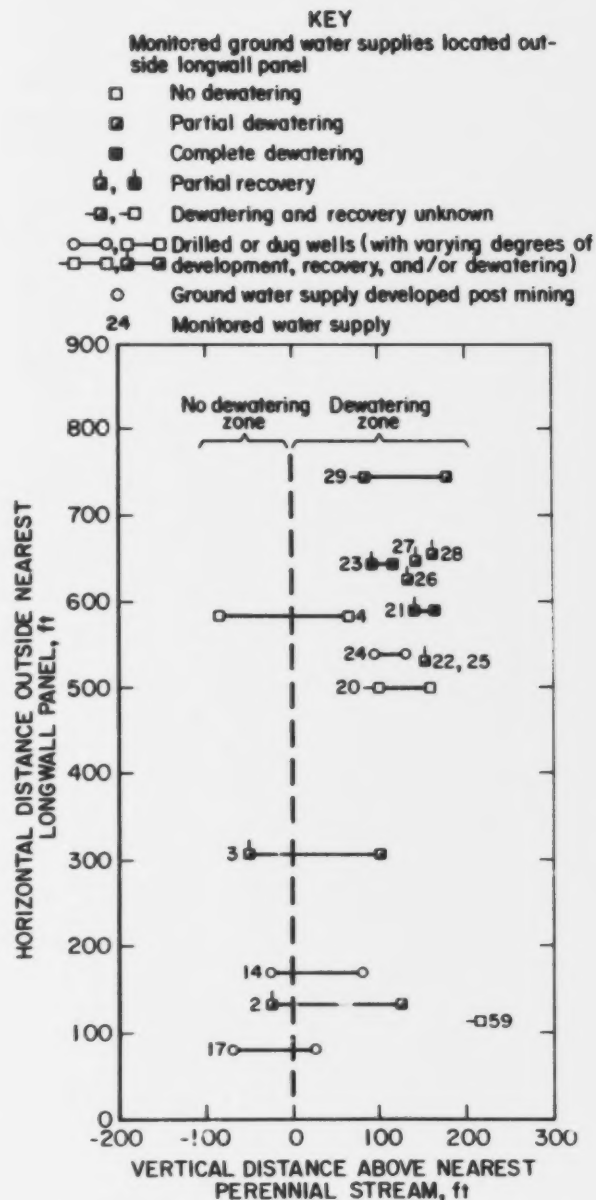


FIGURE 9.—Extent of dewatering for ground water supplies not over longwall panels with respect to their vertical position above the nearest perennial stream at mine site Y.

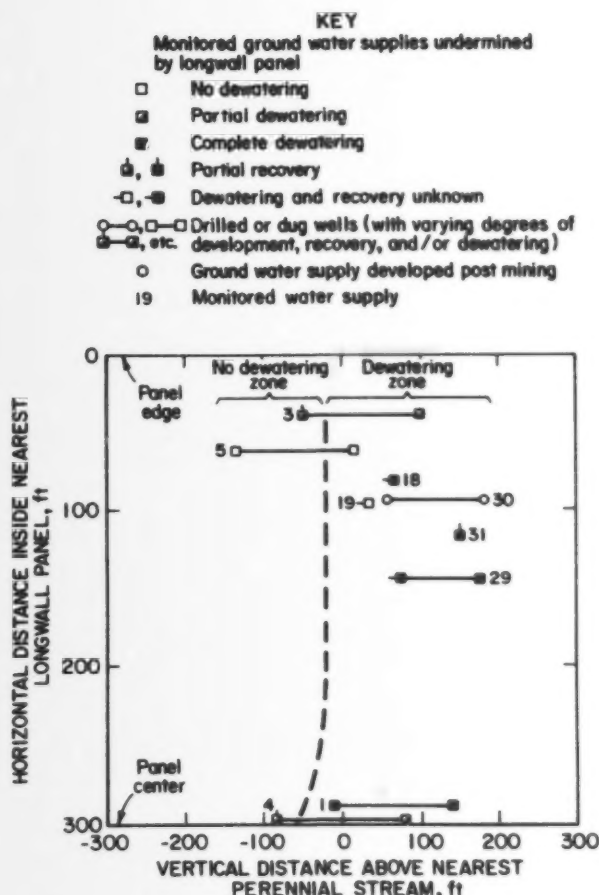


FIGURE 10.—Extent of dewatering for ground water supplies above longwall panels with respect to their vertical position above the nearest perennial stream at mine site Y.

The previous results and supply dewatering trends show that zones of longwall mine subsidence have been associated with extensive strata dewatering extending down to or below the levels of nearby streams but not quite to regional base level as defined by stream H. This probably indicates that increased vertical hydraulic conductivity associated with subsidence fractures has allowed an extensive general lowering of ground water levels to occur under hills and ridges.

The ultimate fate of this drained ground water will be addressed later.

Streams

Measured streamflow data were next examined for any dewatering trends associated with the underground mine. The positions of the streams and their monitoring points are shown in figures 2 and 3, and the stream discharge on June 6, 1986, as a function of watershed area for the subwatersheds is plotted in figure 11. The stream A watershed was divided into three subwatersheds defined by stream B downstream to point B1, stream C downstream to point C1, and stream A downstream of points B1 and C1. Subwatershed B1 was monitored at three different locations on June 6, 1986. At monitoring point B3, streamflow appeared to be normal. The stream discharge to watershed area ratio of 0.062 gal/(min·acre) was within the normal range of median streamflow to watershed area ratio, 0.050 to 0.088 gal/(min·acre), as shown in figure 11 for the streams not impacted by mining. In contrast, subwatershed B2-B3 appeared to have lost water. Whereas stream discharge at monitoring point B3 was 4.75 gal/min, stream discharge at the downstream monitoring point B2 was only 2.00 gal/min. The decrease in stream discharge from B3 to B2 was reflected in the stream discharge increase to watershed area ratio of -0.688 gal/(min·acre), which was about 0.753 gal/(min·acre) less than normal (fig. 12). At monitoring point B1, the stream bed was completely dry. The calculated stream discharge increase to watershed area ratio of stream B1-B2 was -0.667, or about 0.733 gal/(min·acre) less than normal (fig. 12). Subwatershed C1 was completely dry on both June 6 and July 24, 1986, and no measurable streamflow was apparent at monitoring points C2 and C3 on July 24 as well.

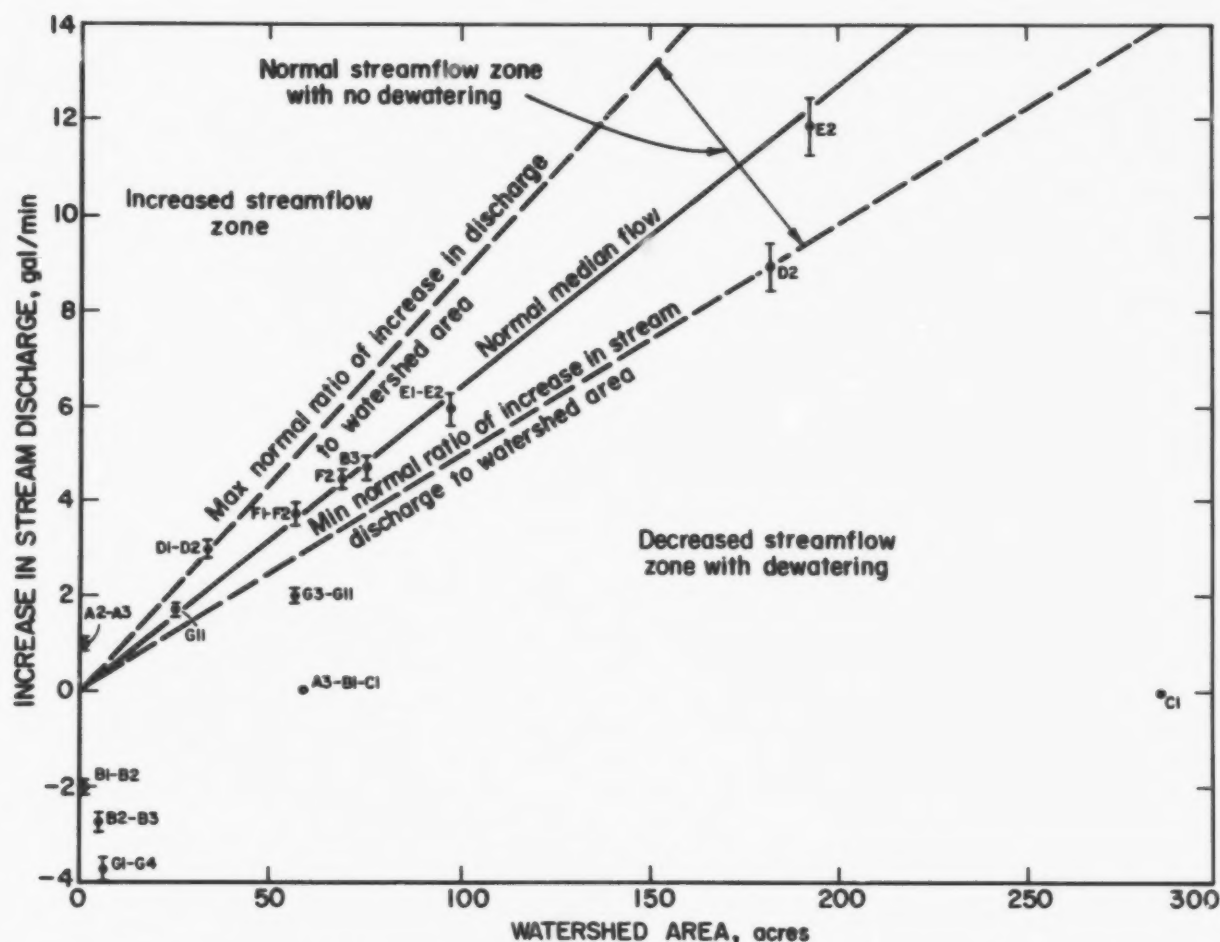


FIGURE 11.—Stream discharge increase versus sub-watershed drainage area for baseflow of June 6, 1986, for control and mining affected streams at mine site Y.

Subwatershed A1-B1-C1 was measured at three points on July 24, 1986. Monitoring point A1 was located 10 ft upstream from its junction with stream H. Monitoring point A2 was located 44 ft upstream from A1, and monitoring point A3 was located 50 ft upstream from point A2. A portion of the A1-B1-C1 watershed, A3-B1-C1, was completely dry when monitored June 6 and June 24, 1986. Directly downstream from point A3, the Jollytown Sandstone crops out in the streambed. Between this outcrop and monitoring point

A2, water was pooled with no measurable streamflow on July 24. In contrast, this stream section exhibited a high streamflow to watershed area ratio of 0.714 gal/(min·acre) on June 6, with flow starting farther upstream immediately downstream from point A3 during the less dry conditions. Stream segment A1-A2 had a measurable streamflow of 5.40 gal/min on July 24, with a calculated stream discharge to watershed area ratio of 2.08 gal/(min·acre). Streamflow at point A1 was not measured on June 6.

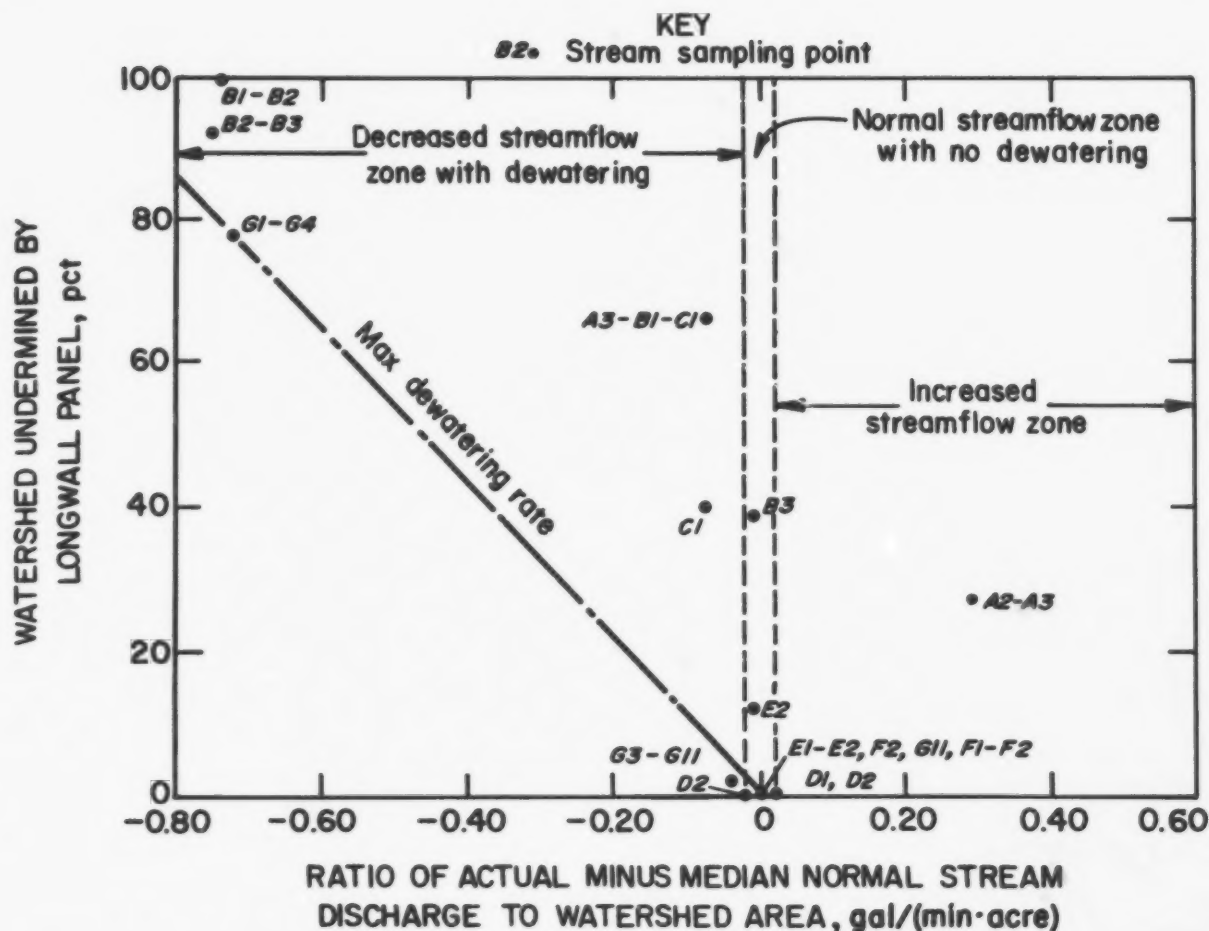


FIGURE 12.—Ratio of actual minus median normal stream discharge to watershed area versus percentage of watershed undermined by longwall panels at mine Y.

Stream G (fig. 3) was monitored on June 6 at 4 sites and on July 24 at 10 sites to test for a possible angle of influence effect on this watershed from panel 3; see figures 12 and 13 for plotted stream data for June 6, and July 24, respectively. Figure 13 shows stream segment G1-G2 near stream H to have been completely dry on July 24. Upstream, segment G2-G3 had pooled water with no measurable flow. Between stations G3 and G9 the stream appeared to have been steadily losing water as it flowed downstream toward stream H. As shown in figure 13, there was an

apparent angle of influence of 31° , as measured from the panel end to point G9 where streamflow had peaked for stream G. Stream G lost water in this 31° dewatering zone as well as in the 21° zone located inward from the panel end (fig. 13). Upstream from station G9 and beyond the influence of panel 3 there appeared to be normal streamflow.

Stream H, the major stream in the mined study area, was monitored on July 24 at two locations. Monitoring station H1 was located downstream from the area of longwall mining, and monitoring station H6

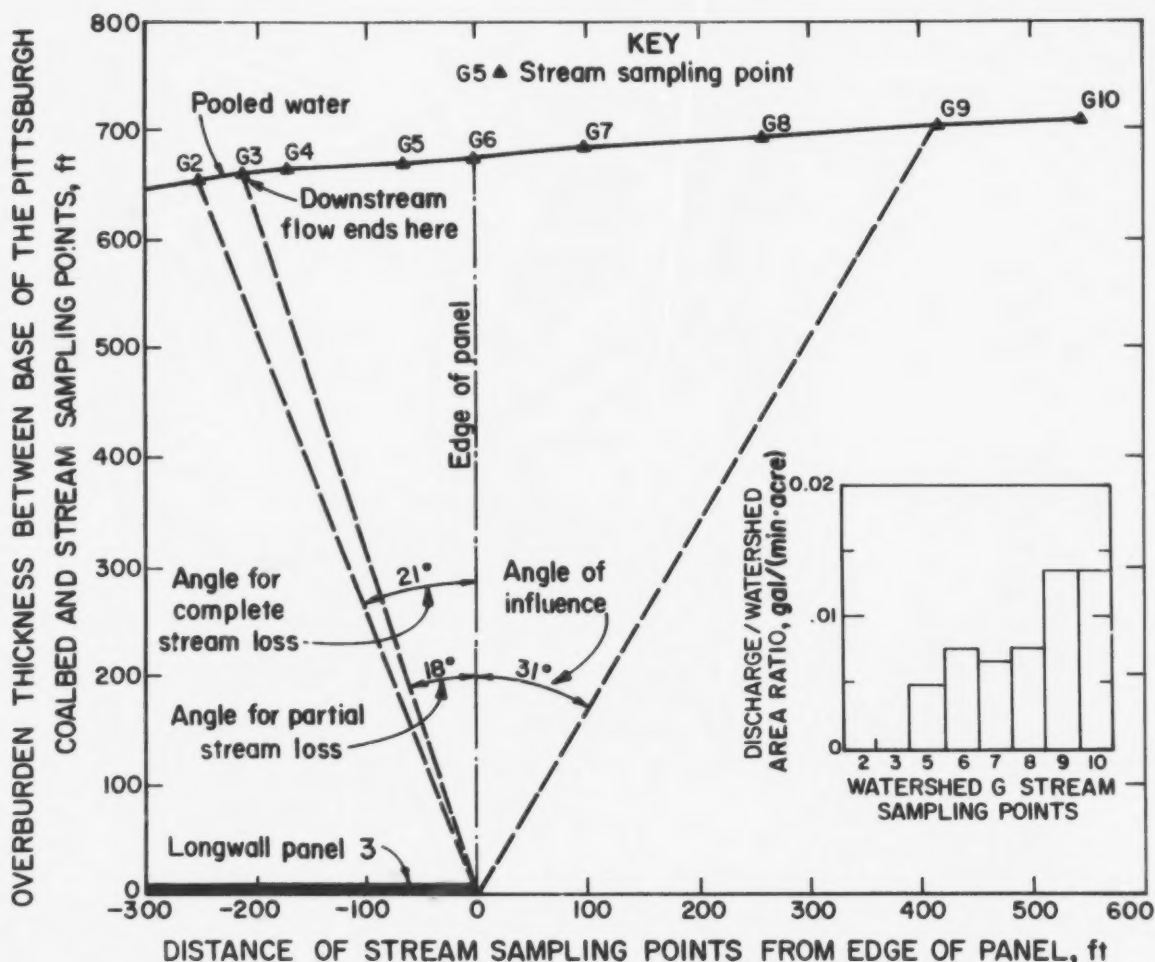


FIGURE 13.—Discharge of stream G as a function of location with respect to the end of panel 3 at mine Y.

was located upstream from the area of longwall mining; see figure 2 or 3. Stream segment H1-H6 was calculated to be gaining 44.0 gal/min. To determine any possible impacts of mining on stream segment H1-H6, its flow rate was compared to that of stream E1, a control stream that was not impacted by mining, as shown by figures 11 and 12. Control stream E1 was selected for comparison to stream segment H1-H6 over control stream G10, which was also not affected by mining, because stream E1 is closer in size of drainage

area to segment H1-H6 and because it has a more similar geologic setting to segment H1-H6. The stream discharge to watershed area ratio at monitoring point E1, was 0.024 gal/(min·acre) on July 24. This value was then multiplied by the watershed area of subwatershed H1-H6 to yield an expected normal stream discharge of 26.3 gal/min. The difference between the measured stream gain of 44.0 gal/min and the 26.3 gal/min represented an excessive streamflow gain of 17.7 gal/min between monitoring points H1 and H6.

This excessive streamflow indicates that subwatershed H1-H6 had a 67 pct greater streamflow than expected compared to the "normal" E1 watershed. Although natural geologic differences may account for some of this difference in streamflow related to ground water runoff contributions, most of this extra flow was probably due to the influence of mine Y.

See figure 14 for the projected stratigraphic locations of these monitored stream points for the mine Y area. This figure is similar to figure 7, discussed earlier. Although stream station E1 could not be plotted on figure 14, it is located stratigraphically below the Jollytown Sandstone.

Fate of Lost Water Over the Mine

The schematic vertical cross section (fig. 15), which is based on dewatering evidence collected from water wells, springs, and streams at mine Y, illustrates a generalized lowering of the

potentiometric surface in response to the longwall mining. The dewatering data indicate that the water lost from the surface to near-surface zones was recharging down to the Jollytown Sandstone or adjacent strata located immediately above the regional base level. This probably occurred because of tensional subsidence fractures in the hills which had significantly increased the vertical permeability and facilitated the rapid downward drainage of water. Although no obvious fractures were observed in the soil zone above mine Y, mine subsidence theory nonetheless predicts their presence in bedrock of an upper fractured zone or zone of surface rock fracturing (8). This lost water could have then possibly flowed downdip along the Jollytown Sandstone to discharge into major streams in the local area.

There does not appear to be any evidence that these lost waters recharged to the underground mine. Evidence against such deep mine recharge is the probable

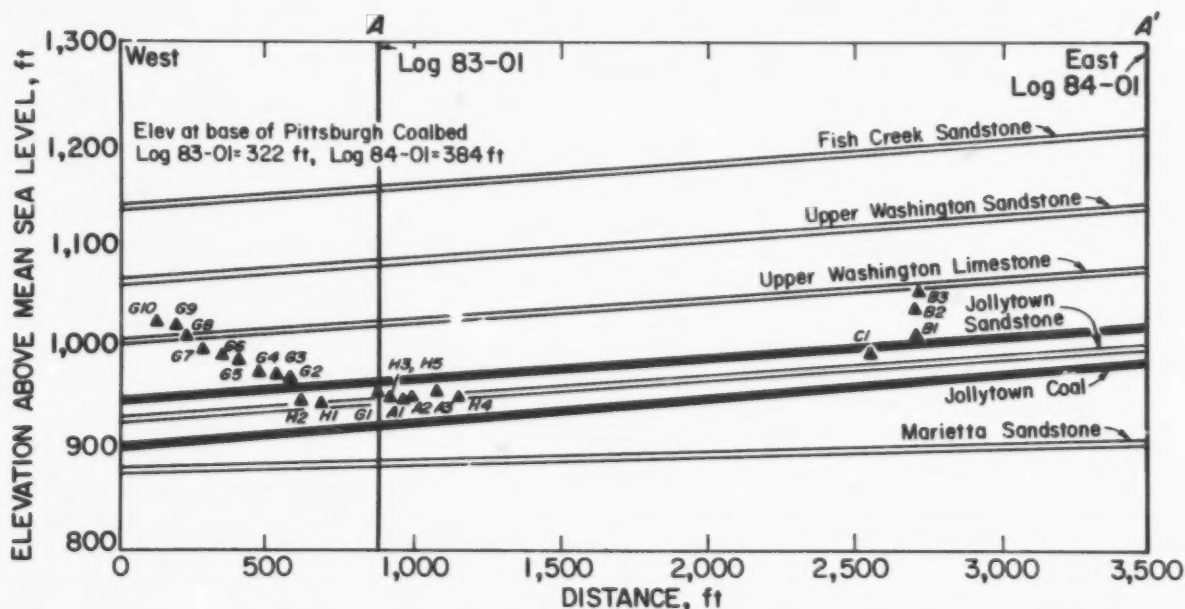


FIGURE 14.—Location of selected stream monitoring stations with respect to stratigraphic cross section A-A' at mine Y.

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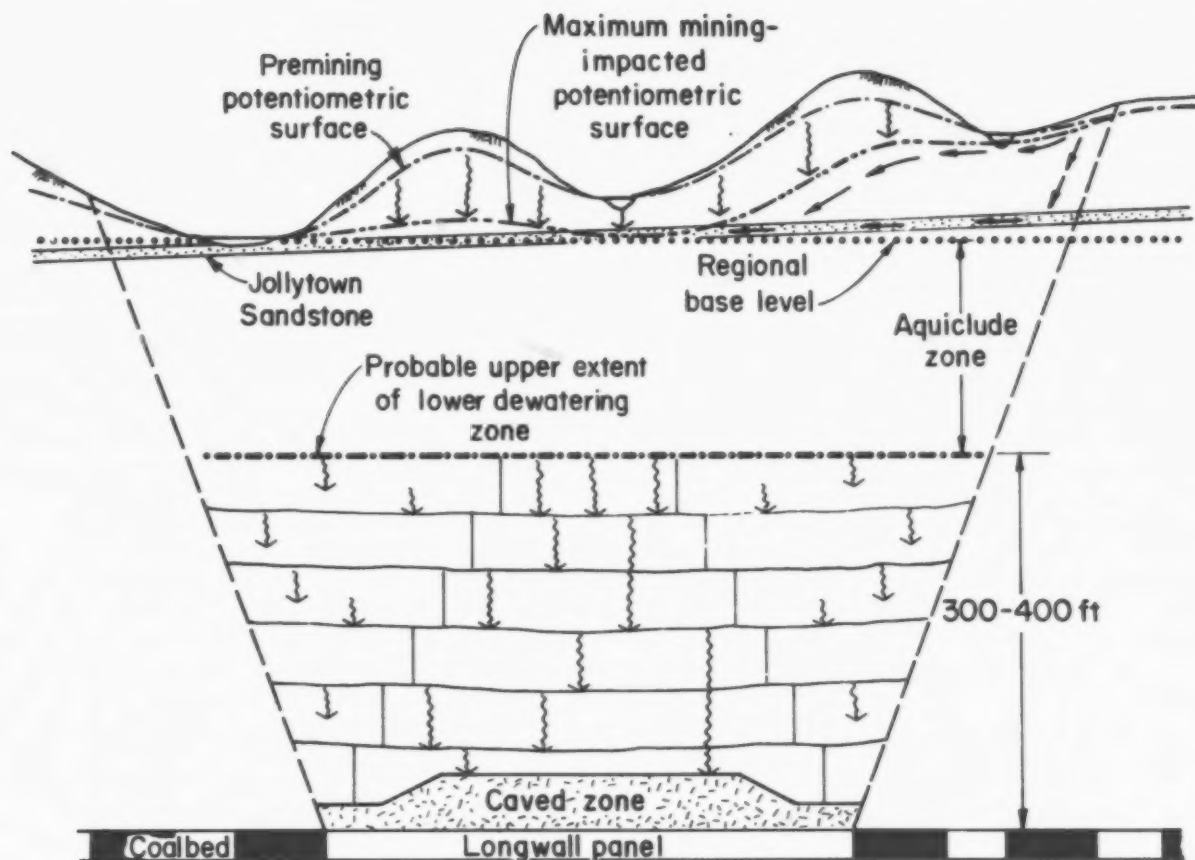


FIGURE 15.—Schematic vertical cross section showing generalized dewatering effects at mine Y.

presence of an aquiclude zone located below the regional base level (fig. 15). That zone is predicted by mine subsidence theory, since the minimum mine overburden thickness (500 ft) far exceeds the probable vertical extent of the lower fractured and cavied zone [30 to 60 times the mined coal seam thickness (8)]. Also, the greater lateral compressive stress and elastic rebound of subsided strata within the aquiclude zone should keep fracture permeability relatively low there (8). Lack of significant vertical movement of water through this zone is indicated by several nondewatered wells within it and the formation of five new springs just above it. Additional evidence against deep mine recharge is that company personnel at mine Y reported that this mine received only about 0.01 gal/(min·acre) of recharge.

Stream H was investigated as a likely major stream to be receiving lost water runoff. The apparent excessive increase in streamflow for stream H must be accounted for by the inclusion of an additional water source that is not normally a part of the stream's hydrologic budget. Otherwise, stream H would have had a normal stream discharge to watershed area ratio, which would have been typical for an unmined stream in the area. The most probable additional water source was extra water saved from evapotranspiration due to the effect of mining.

To determine the possibility of an evapotranspiration saving accounting for the increased streamflow at stream H, it was first necessary to estimate the evapotranspiration rate (ET) for the study area. The U.S. Geological Survey (Pittsburgh office) has reported average

precipitation to be 39.4 in/yr for the period 1941-80 based on a rain gauge at Waynesburg near mine Y (5). They also estimated that ET averaged 61 pct of total precipitation for this same period for a portion of the watershed of South Fork Tenmile Creek that includes mine Y (5). Based on these data, the ET should have averaged 24.0 in/yr for the 1941-80 period at mine Y. This is equivalent to 6.52×10^5 gal/(acre·yr). That ET value was then multiplied by the 535 acres of longwall mine area (including the zone of the 42° angle of dewatering influence) that could have been contributing to ground water runoff entering stream H. This produced a yield of 3.49×10^8 gal/yr, or 663 gal/min, of average ET from the longwall mine area. The ratio of excess streamflow in the measured stream H segment (17.7 gal/min) to the average ET over the mine is 2.7 pct. Therefore that excess streamflow could be accounted for by a 2.7-pct saving in ET over the mine. This water saving could have been realized following subsidence by a diversion of shallow ground and stream waters along subsidence fractures to deeper levels closer to the regional base level, where the water would be less susceptible to evaporation or transpiration by plants. The water that was thereby spared from ET would have been added to the ground water runoff portion of streamflow for stream H.

WATER RECOVERY FOLLOWING MINING

Some recovery of water levels occurred for domestic water supplies and streams following mining at mine Y. Of the two accessible water supplies that had been partly dewatered over longwall panels, both showed partial but not complete recoveries. Of the four accessible supplies that were completely dewatered over longwall panels, only two had a partial recovery and none had a complete recovery. Of nine affected water supplies for which recovery data were available and which were not undermined by a longwall panel but were located within the zone of the 42° angle of influence, eight had a

partial recovery and one had a complete recovery. Water supplies that showed partial to complete recoveries did so within 1 to 3 yr after dewatering occurred and were typically located close to or below nearby perennial streams or hydrologic base level. Supplies located over panel centers showed less overall recovery compared to supplies located near panel edges or adjacent to panels. Only one supply (well 2) has exhibited complete recovery so far.

While some old affected springs showed partial recovery, several new springs appeared or were developed at lower elevations following mining. Two new seasonal springs were developed over panel 2; one new perennial spring was developed over panel 3 and is being used to supply five families; one new seasonal spring was developed over panel 4; and one new spring was developed over panel 5. All of these springs appeared immediately to 1 yr after undermining occurred. The new springs appearing above panels 2 and 3 are located stratigraphically at the top of the Jollytown Sandstone, whereas the new springs above panels 4 and 5 are situated approximately 20 ft above the stratigraphic level of the Jollytown Sandstone (fig. 14). In addition, a domestic well that was not dewatered over the mine had a reported increase in yield following mining; this well penetrates to below the Jollytown Sandstone. These enhanced water supplies again indicate that overlying aquifer waters were diverted downward and that partial recovery of ground water often occurred close to the local or regional base (stream) level within 1 yr of mining.

A comparative analysis of streamflow to watershed area ratios for June showed indirectly that panel 1 had a complete recovery and panels 2 and 3 had partial recoveries since longwall mining occurred. The portion of panel 1 that undermined subwatershed B3 was between 3.0 and 3.5 yr old, whereas the portion of panel 2 that undermined subwatershed B2 to B3 was between 2.0 and 2.5 yr old. Subwatershed B2 to B3 was calculated to be losing 0.811 gal/(min·acre of longwall panel) to

the subsurface. In contrast, the portion of panel 3 that undermined the stream G watershed was between 1 and 2 yr old, and subwatershed G4 to G11 was calculated to be losing 2.08 gal/(min·acre of longwall panel) to the subsurface. No calculations of maximum streamflow loss, such as those above, were possible for

panels 4 and 5 owing to the dry streams encountered over these panels. The obvious trend from these data is that streams show progressive recovery with time between about 1 and 3 yr of longwall mining and recover their normal premining flow by about 3 yr after mining.

CONCLUSIONS FOR MINE Y

1. Room-and-pillar mining (with no subsidence) has not affected water supplies or streams.

2. Above longwall panels, 8 of 11 water supplies appeared to have been partly to completely dewatered. Dewatering occurred in a zone about 655 ft and higher above the mined Pittsburgh Coal Seam, but not closer to this seam. Dewatering was greatest above the center of the longwall panels. All supply dewatering over longwall panels occurred within a range of 0.75 to 1.00 for the ratio of longwall panel width to mine overburden thickness.

3. Off of and away from the longwall panels, 11 of 13 water supplies were partly to completely dewatered within a maximum angle of dewatering influence of 42° from the nearby panel ribs or ends. Dewatering occurred in a zone of about 720 ft and higher above the mined Pittsburgh Coal Seam, but not within 720 ft of this seam.

4. Dewatering did not occur below the regional base level, as represented by the major stream over the mine. Water supplies developed within 800 ft laterally of the major stream and no higher than 50 ft above it were not dewatered, but such supplies over 50 ft above it were partly to totally dewatered.

5. Water supplies developed at or below the elevation of the nearest perennial stream and outside of panels but inside the 42° angle of influence zone did not appear to be dewatered. However, such supplies developed mostly above the level of the nearest perennial stream were at least partly dewatered. Most water wells developed more than 20 ft below the elevation of the nearest perennial stream and undermined by longwall panels

were not dewatered, but such wells located above this level were partly to totally dewatered. Above the center of one longwall panel, a well was dewatered to a depth of 60 ft below the elevation of the nearest perennial stream but not below regional base level.

6. Streams located above regional base level and undermined by longwall panels less than 2.5 yr old were partly to completely dewatered during baseflow conditions. An angle of dewatering influence of 31° existed outside the end of such a panel for one measured stream, with partial stream dewatering having occurred within this area adjacent to the panel. Streams located above regional base level and also above panels at least 3 yr old had normal flows.

7. Water from lost ground water supplies and streams did not penetrate to the mine, but instead migrated downward through probable subsidence fractures to near regional base level, where it migrated laterally through the Jollytown Sandstone to finally discharge to the largest area stream over the mine. Reported ground water recharge to the deep mine was only 0.01 gal/(min·acre). The major area stream flowing over the mine received about 17.7 gal/min more than the expected normal runoff contribution on July 24, 1986. This extra slug of runoff during dry baseflow conditions was most likely contributed by a potential evapotranspiration reduction of about 2.7 pct over the mine that resulted from reduced stream and ground water levels at higher elevations.

8. Several streams and water supplies showed partial to total recovery following mining effects. Streams appeared to

have had complete recovery within 3 yr after longwall mining occurred. Of the accessible ground water supplies over longwall panels, all partly dewatered supplies had partial recovery, but only one-half of the completely dewatered supplies had a partial recovery, with no complete recovery observed. Of the affected accessible supplies adjacent to longwall panels, all showed at least partial recovery. Several new springs also formed close to stream level within 1 yr

of nearby mining. Any noted recovery for affected wells and springs occurred within 1 to 3 yr of the initial mining effects, with more extensive and rapid recovery for supplies near stream level or not over panel centers. Only one affected water supply had shown complete recovery as of February 1986. Affected streams exhibited more complete recovery than did wells and springs following mining.

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